BLOCK CAVING AT KELLEY MINE, THE ANACONDA COMPANY, BUTTE, MONT.

BY C. C. POPOFF

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by

C. C. Popoff 1/

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2/ All illustrations are courtesy of The Anaconda Company.

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PART I - GENERAL

INTRODUCTION AND SUMMARY

This is one of a series of reports dealing with mining methods and performance of mines in the United States to be published by the Bureau of Mines. A similar program was conducted from 1928 to 1939 that was generally conceded to have contributed substantially to improving the efficiency of mining operations. The publications provided practical technological data for use by technical educational institutions and source material for engineering handbooks and mining engineering textbooks.

The primary purpose of the program is to promote the development and conservation of our mineral resources by disseminating technological information on methods, practices, and related features in mining mineral deposits. It is expected that the program will (1) promote efficiency in operating mines, (2) assist operators and engineers in evaluating and developing new mineral deposits, (3) provide data for engineering curriculums in mining schools, (4) provide up-to-date factual performance and cost data for the use of research agencies of the Government in preparing reports for use in developing mineral policy, (5) furnish performance and cost data for revising and up-dating engineering handbooks, and (6) for inclusion in engineering textbooks.

A product of the program will be summary bulletins covering broadly related mining operations and presenting their applications and limitations under varying conditions, with discussions of the problems encountered and their solution. Summary reports will also review foreign and domestic research activities within related fields and present briefs of theoretical studies applicable to the solution of mining problems. The ultimate goal is to increase and disseminate to the industry the fund of technical knowledge that may be applied to the engineering analysis and solution of mining problems.

This report describes block caving at the Kelley mine, owned and operated by The Anaconda Co. at Butte, Mont. (figs. 1 and 2). The deposits, engineering work performed in the introduction of this method, problems encountered, and mine facilities are discussed in part I. The principles used in design and development and standard operative procedures are presented in part II.

The successful application of block caving at the Kelley mine extends the use of this method to wide and steeply dipping zones of low-grade copper mineralization. These zones had been partly stoped to extract high-grade ore and left open, timbered, or rock- and slime-filled.

The shape and size of minable zones were determined by underground exploration. Elaborate evaluation included channel sampling of ore exposed in workings, mine-car sampling, and bulk sampling of all material excavated; the latter were for extensive metallurgical testing and for determining factors for correcting mine assays.

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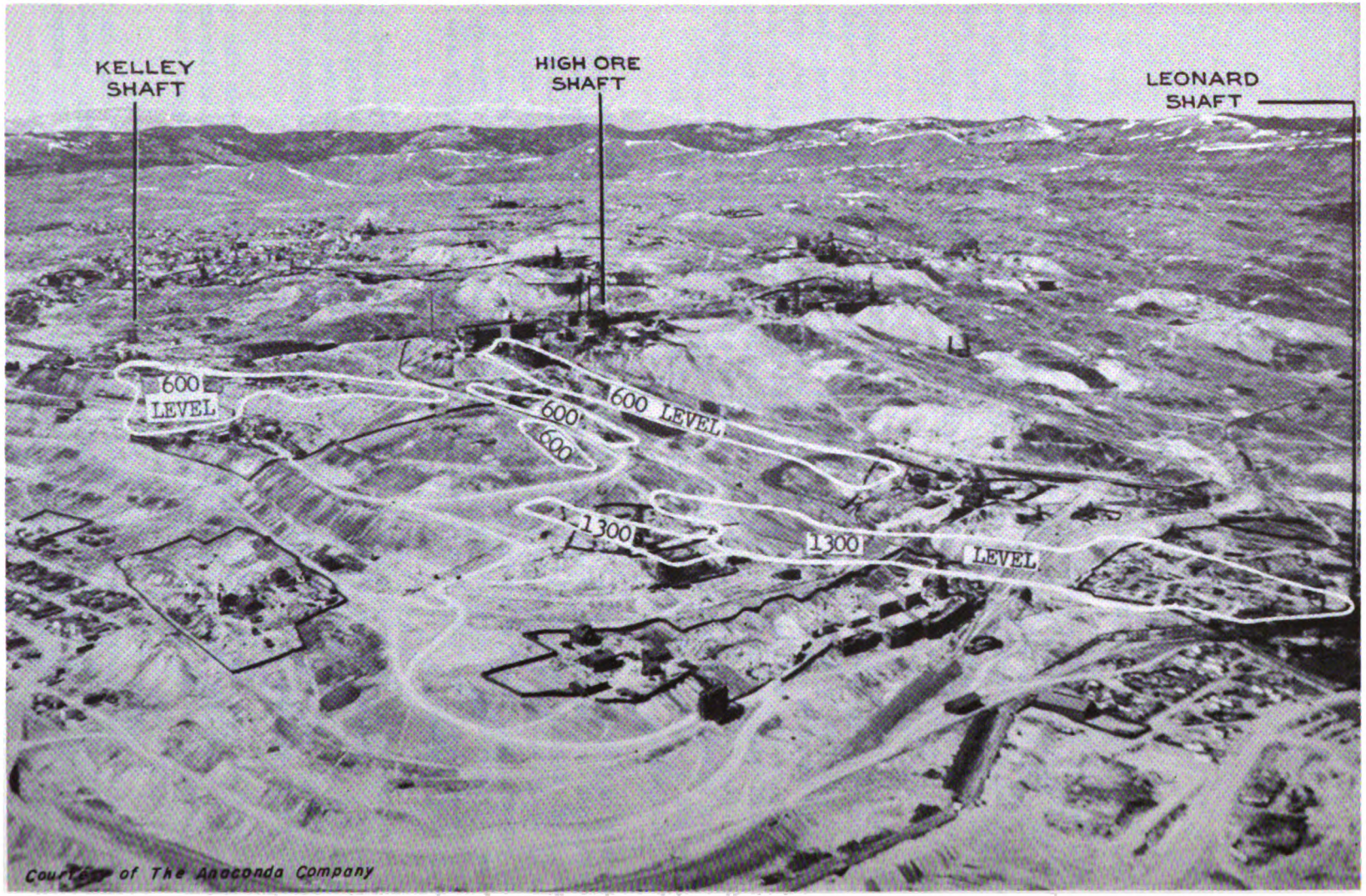

Figure 1. - Aerial view of Kelley mine area with ore zones projected to surface.

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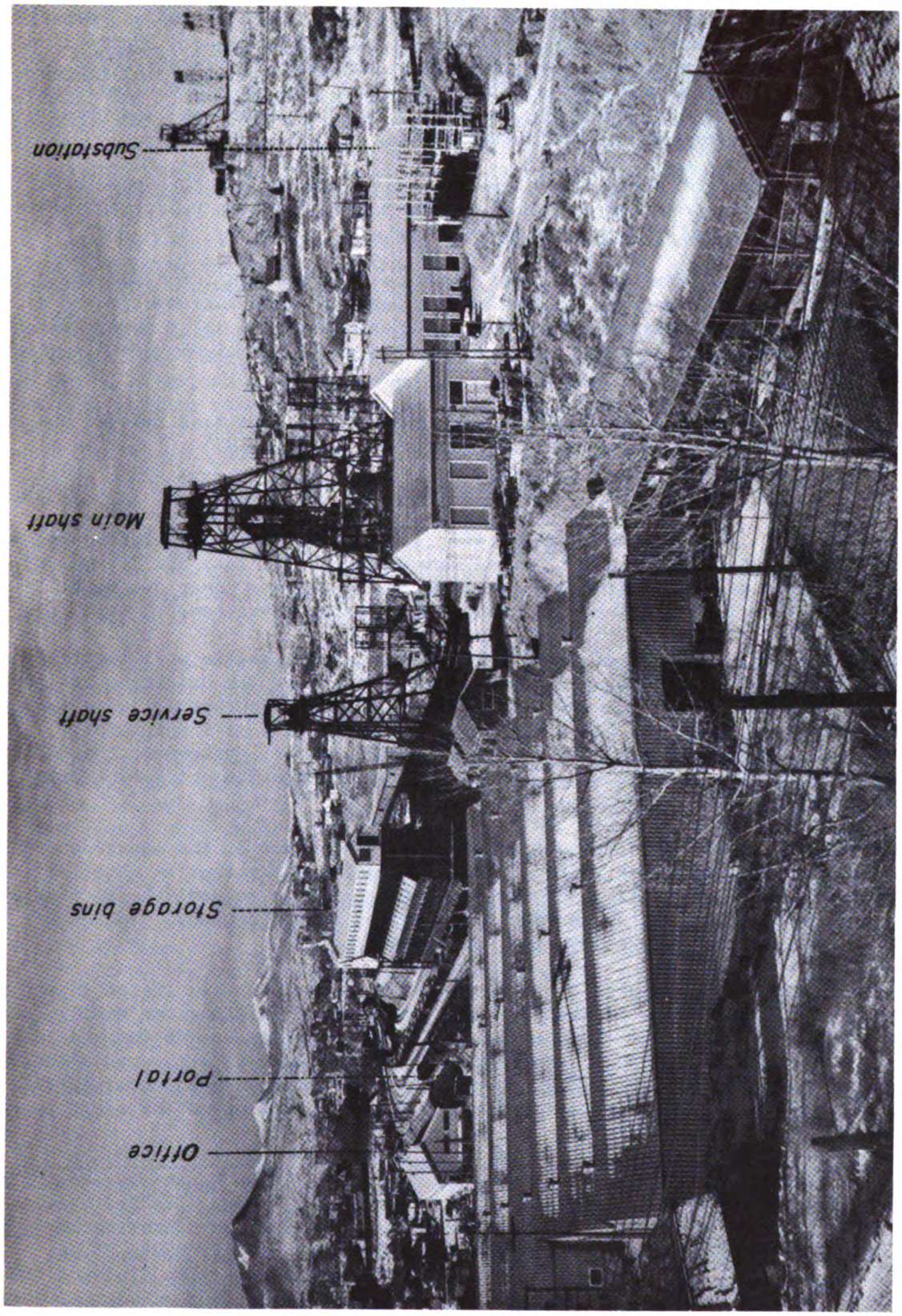

Figure 2. - General view of Kelley mine and surface installations.

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Block caving at the Kelley mine was adopted after careful study and experimental mining. Results were applied in mine design and in working out standard procedures for all mining operations. Many intricate problems were encountered, such as underground dormant fires and gas and timber and junk iron in old workings. These were overcome by careful planning, by developing new techniques and equipment, and by constant engineering control and supervision.

After initiation of the project, production experience and continued efficiency studies of both operations and equipment by the Mining Research Department of the company prompted several changes and innovations.

Originally an all-gravity system of block caving was used. The successful utilization of concrete to support development workings made introduction of a slusher-gravity system possible, first on an experimental and later on a production basis. Other changes to increase safety and the overall efficiency are discussed in this report.

Although considerable difficulties were encountered, the mine reached the full capacity of 15,000 tons a day output, as scheduled, in July 1955.

ACKNOWLEDGMENTS

Introduction and advance of the Greater Butte block-caving project to its present stage are due to the foresight of the management of The Anaconda Co. and to the engineering skill and team effort of the staff and engineers under the direction of Chester H. Steele, vice president in charge of western operations.

The project is a result of more than 60 years of systematic work compiling and appraising geologic, sampling, and mining data that proved the existence of large low-grade copper zones not minable profitably by any other method; of wide experience in mining, including 40 years of block caving in mines at Inspiration, Ariz., [3 4 5] and the Andes in Chile, South America; and of systematic consideration of local conditions, application of new ideas, and comparative study of mining operations and costs.

The author acknowledges the cooperation and assistance of the management and of the engineering and operating staff of The Anaconda Co. in gathering material for this report. He expresses his appreciation for permission to publish the maps and photographs in this report, which were all supplied by the company, and for the opportunity to inspect the Kelley-mine operations.

Several published articles about Butte mining operations are listed in the report. A series of papers presented at the meetings of the American Mining Congress and the American Institute of Mining and Metallurgical Engineers during the last 2 years by individual engineers of The Anaconda Co., although as yet unpublished, was made available to the author. These papers were freely drawn upon and, together with lengthy discussions with the members of the engineering and operating staff, were invaluable in preparing this report.

3/ Lehman, G. R., Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.: Trans. AIME, vol. 55, 1917, pp. 218-231.

4/ Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

5/ Stoddard, A. C., History and Development of Block Caving at the Mines of the Inspiration Consolidated Copper Co.: Trans. AIME, vol. 163, 1945, pp. 96-121.

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HISTORY

Since the discovery of silver-gold lode deposits in 1864 the Butte district has produced over $2,250,000,000 worth of metals, an unprecedented value in the mining world. The principal mineral land owner and mine operator since 1895 has been the Anaconda Copper Mining Co., now named The Anaconda Co. This company and its subsidiaries form an integrated enterprise that includes the mining and smelting of copper, zinc, lead, manganese, gold, and silver, the recovery of various minor metals, and the manufacture and distribution of metal products.

The history of mining at Butte is that of constant search for increased efficiency by using new mining methods, mechanizing mining operations, improving organization, and bettering working conditions and safety.

The Anaconda Co. was first to introduce wet drilling and sprinkling to prevent silicosis, use electric cap lamps, control ventilation mechanically, condition air by taking advantage of local atmospheric conditions, cool and clean underground air near the working faces, and have an organized campaign of accident prevention. It was one of the first mines to develop detachable bits and one of the first to use power-driven scrapers, mechanical mucking machines, high-speed electric-motor-driven blowers, and canvas tubing. The company was the first in metal mining to include a ventilating engineer on its engineering staff.

Before the Greater Butte project, development and mining in the district had been carried out through about 55 different shafts, 1,000 to 4,000 or more feet deep, having a maximum size of 6.75 by 20.3 feet, all timbered, and with capacities of as much as 2,000 tons a day.

For the last 40 years extraction of ore from Butte veins and ore shoots has undergone drastic changes. On the whole, these changes have followed advanced ideas and greatly contributed to better underground practice throughout the country.

Before 1912 about 95 percent of the ore was extracted by overhead stoping, with square-set timbering. Later, backfilling was used in wide ore bodies where the back was considered unsafe and in ground where ore and walls were too weak for open-stope mining, with or without stulls.

Overhead, inclined cut-and-fill or rill stopes have been used since 1923 in places where the walls were fairly hard and the vein was narrow. Until later developments rill stopes were considered the most economical and were used extensively. Some rill stopes were timbered by square sets when the back was unsafe for stoping, as in wide veins, or where ore and walls were not strong enough to stand without support.

Since the introduction of mechanical scrapers, a horizontal cut-and-fill system has been employed to meet specific conditions, as in veins dipping less than 50°. The modified Mitchell top-slice system has been successfully tried for broken and heavy blocks in fairly wide ore bodies or for pillar extraction.

Shrinkage stoping, in which broken ore is accumulated until the stope is completed, was tried, but found inflexible; it permitted no sorting, caused dilution, and tied up production until a stope was completed.

An outstanding stoping development occurred when a complete changeover was made from conventional square-set system to vertical-slice stoping for mining narrow veins and to slot stoping across the wide veins in 1941-44. Percentages of production

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obtained by various methods of stoping in the Butte district in the latter half of 1944 are listed below:6/

Stoping method	Type	Percentage
(classification as used by The Anaconda Co.)		
Square set	Conventional	33.4
	Vertical slice	23.1
	Slot	25.6
	Timber rill	.4
	Mitchell slice	.9
	Subtotal	83.4
Cut-and-fill	Horizontal	12.5
	Inclines (rill)	4.1
	Subtotal	16.6
	Total	100.0

BLOCK CAVING - GENERAL FEATURES

The block-caving method of mining consists of dividing suitable ore bodies into blocks of predetermined size and undercutting each from the bottom and, when necessary, from the sides to permit rock stresses to cave and crush the ore into fragments that can be easily handled. The method is applicable to homogeneous and rather weak ore bodies of regular outline with enough horizontal area to cave freely; strong walls and capping or overburden that cave freely are required also.

The method utilizes the instability of the ore for caving, shear and compressive stresses for crushing, and gravity for moving the ore to drawpoints.

Block caving is characterized by: (a) Descending order of mining levels; (b) ore drawing of each level from bottom to top; (c) surface subsidence.

Block caving is amenable to mass production and is an efficient and low-cost mining method permitting scientific planning, centralized management, labor specialization, and modern mechanization. The successful use of this method requires detailed engineering study of the nature of an ore body by systematic and detailed exploration and sampling. Adequate development, preparatory work, transportation, ore drawing, and allied mining operations are equally important.

In the United States use of this method is widely accepted for mining.7/

BLOCK CAVING AT BUTTE

The Greater Butte project, one of the major postwar mining enterprises in the United States, was announced by the company in September 1947 after careful evaluation of future domestic copper requirements and the world copper market. This project was initiated after a comprehensive and careful appraisal of available geologic and mining information and a considerable amount of preparatory work that included rehabilitation of old workings, driving new exploratory openings, elaborate sampling

6/ Bishop, L. F., Vertical Slice and Slot Stoping at Butte: Trans. AIME, vol. 163, pp. 262-273.

7/ Bucky, P. B., Mining by Block Caving. (A series of articles reprinted from the Explosives Engineer): Pub. by Hercules Powder Co., Wilmington, Del., 1945, 104 pp.

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(1943-46), metallurgical testing (1943-1947), and experimental block caving (1945-46).[8 9/] After necessary information with regard to ore distribution, and the shape, dimensions, grade, and physicochemical characteristics of individual ore zones had been determined, a decision was made to mine the ore at an ultimate rate of 15,000 tons a day at an estimated cost that would assure profitable operations.

Planning and actual operations encountered many problems unknown in other block-caving mines. Execution of the project required (a) the most comprehensive appraisal of all geologic, engineering, and economic factors concerning utilization of low-grade ores; (b) one of the most complicated, detailed, long-range planning and designing programs of any new mining enterprise; and (c) incorporation of the new project into the existing mining facilities and operations without disturbing the output of active mines.

In general, the high-grade veins and ore shoots mined by the square-set method had resulted in stopes only partly filled with submarginal ore or waste. Filling the remaining openings with wet mill tailings was an important preliminary step to block caving.

Mine development of the upper levels included sinking and raising the main and auxiliary shafts of the Kelley mine, driving main haulageways on the 600 level, preparing ore-storage bins and skip-loading facilities on the 700 level, and driving intermediate haulageways on the 400 and the 500 levels.

Mine development of lower levels is continuing by deepening the main shaft and by driving horizontal workings.[10 11 12/] Haulageways on the 1,300 level, ore-storage bins and skip-loading facilities on the 1,400 level, and haulageways on the intermediate 800 and 1,100 levels are advanced to permit mining ore from the second lift.

The greater part of block development and ore drawing is concentrated on the 600 level.

Total capital expenditure for the Greater Butte project exceeds 30 million dollars.

PRODUCTION BY BLOCK CAVING

Ore drawing began in April 1952 and reached the planned capacity of 15,000 tons a day during July 1955.

In the first year of block caving the mine produced about 1.1 million tons of ore, averaging 1.07 percent copper recovered. This output equals 53.3 percent of the ore tonnage mined underground and 19.9 percent of the copper tonnage recovered

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8/ McGlone, E. S., The Greater Butte Project: Eng. and Min. Jour., vol. 149, September 1948, pp. 80-82.

9/ Steele, C. H., How the Greater Butte Project Was Born: Min. Cong. Jour., March 1949, pp. 32-34, 39.

10/ Huttl, J. B., The Greater Butte Project: Eng. and Min. Jour., vol. 152, October 1951, pp. 72-81.

11/ Huttl, J. B., Greater Butte Mine Output Nears 10,000 Mark: Eng. and Min. Jour., vol. 154, April 1953, pp. 77-81.

12/ Huttl, J. B., Anaconda Changes Mining Techniques as Butte Output and Reserves Climb: Eng. and Min. Jour., vol. 156, August 1955, pp. 86-89.

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from copper mines in the district (table 1). In 1954 production from block caving increased to 78.8 percent of the ore mined underground and to 45.9 percent of the copper recovered from copper mines.

The mine is operated two 8-hour shifts plus a skeleton crew for hoisting ore on the graveyard shift 6 days a week. A total of 800 men is employed.

DESCRIPTION OF DEPOSITS

Geology

The complex geology of the Butte mining district has been discussed in several publications.[13] [14] [15] The district, about 6 square miles in area, is characterized by intense fracturing, by alteration of quartz-monzonite country rock, and by concentric zoning of mineralization. In the central zone, north of Butte and west of Meaderville, the mineralization consists of copper sulfides oxidized to varying depths. The depth of mineralization is known to extend for about 1 mile. High-grade ore veins and ore shoots in this zone have been mined since 1864 through many shafts, of which the Anaconda and the Leonard are the most famous.

The Kelley mine's minable low-grade copper ore bodies, locally called ore zones, are in the eastern part of the central zone of mineralization. The high-grade ore occurs in pyrite-enargite veins and ore shoots related to branching and interlacing structures of different ages, which intersect the quartz-monzonite. The most highly productive Anaconda system of tension fractures has a prevailing east-west strike and south dip; the Blue Vein set of horizontal slip fractures strikes northwest and dips southwest. Lastly, the Steward fault veins vary in width from a few inches to a maximum of 50 or 60 feet. Some veins are traced as far as 7,000 feet in length and extend to a depth of several thousand feet. Some veins extend to the surface; some are blind.

The Kelley low-grade ore bodies may be divided into two types. The wide-vein zones consist of closely spaced fissure veins with numerous ore veinlets and stringers traversing the intensely altered quartz-monzonite, making certain parts of it minable by block caving. The other type is related to the horsetail system of southeast-striking fractures, branching from the high-grade east-west veins, that have replacement copper ore minerals localized to a degree suitable for block caving.

Ore Zones

Twelve copper ore zones have been outlined by exploration above the 3,400 level. In general, they may be considered as regular-shaped, steep-dipping (65°-90°) bodies whose boundaries are determined by sampling (fig. 3). In size they range from 2 to 60 million tons. The average grade of minable ore bodies is at least 1.00 percent (20 pounds) of recoverable copper a ton, with minor amounts of silver and gold.

Ore zones 6, 8, 9, and 11, now being mined on upper levels, lie between the 200 and 1,200 levels and are 100 to 200 feet wide and as long as 1,100 feet.

13/ Weed, W. H., Geology and Ore Deposits of the Butte District, Mont.: Geol. Survey Prof. Paper 74, 1912, 262 pp.

14/ Sales, R. H., Ore Deposits at Butte, Mont.: Trans. AIME, vol. 46, 1914, pp. 3-109.

15/ Sales, R. H., and Meyer, C., Preliminary Study of Vein Formation, Butte, Mont.: Econ. Geol., vol. 44, No. 6, 1949, pp. 465-484.

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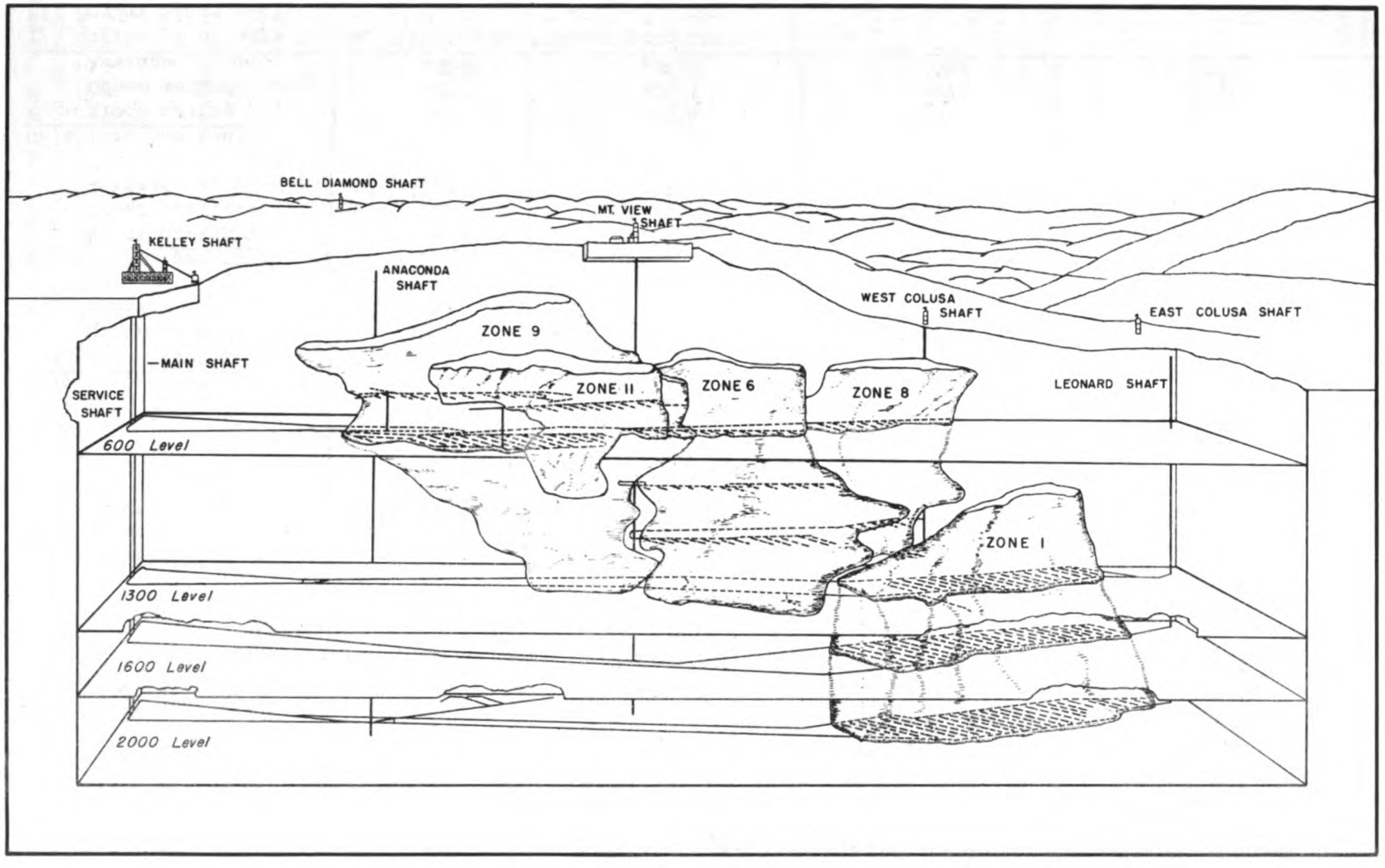

Figure 3. - Block diagram—Greater Butte copper ore bodies.

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TABLE 1. - Production of copper ore and copper metal by block caving compared with other underground methods in Butte district, Mont.

	1951		1952		1953		1954 1/	
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total
Production: Copper ore (dry weight)								
Block caving	-	-	1,095,064	53.3	2,899,565	72.4	1/2,818,708	78.8
Other mines 2/	1,039,511	100	960,108	46.7	1,107,103	27.6	1/757,532	21.2
Total	1,039,511	100	2,055,172	100	4,006,668	100	1/3,576,240	100
Production: Copper metal								
Block caving	-	-	11,738	19.9	29,150	36.4	1/28,830	45.9
Other mines	48,759	100	47,351	80.1	53,031	63.6	1/34,019	54.1
Total	48,759	100	59,089	100	80,181	100	1/62,849	100

Average yield, in terms of recoverable metals

	1951	1952	1953	1954
Copper, percent:				
Block caving	-	1.07	1.01	1.02
Other methods	4.69	4.93	4.61	4.49
Average	4.69	2.88	2.00	1.76
Silver (oz. per ton):				
Block caving	-	.34	.32	.35
Other	1.33	1.26	1.27	1.30
Average	1.33	.77	.58	.55
Gold (oz. per ton):				
Block caving	-	.0026	.0023	.0026
Other methods	.0026	.0025	.0022	.0023
Average	.0026	.0025	.0023	.0025

1/ Strike of 53 days during 1954 caused reduced production.
2/ Copper mines only.

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Ore zone 1 has been explored from the 625 to the 2,200 level - a total height of about 1,600 feet. The zone is 100 to 400 feet wide and 1,600 feet long.

Mineralogy

The quartz-monzonite of the Butte district is composed of feldspar, quartz, and sericitic and flake muscovite, with minor amounts of pyrite and other sulfides. The rock is composed of fine grains that vary from less than 1 to as much as 3 mm. in size. Replacement of quartz and feldspar by sericite is a characteristic alteration. Vein quartz, sericite in interstices, and pyrite along fractures cause the quartz-monzonite to be of nonuniform and rather low compressive strength when compared with granite (table 2).

The Kelley ore consists of altered quartz-monzonite, vein quartz, pyrite, and various minor quantities of copper sulfide minerals - enargite, bornite, chalcocite, and chalcopyrite. All these minerals occur in veins, stringers, and as disseminated grains.

The oxidation of sulfides in the upper levels has advanced to a considerable degree, with the formation of soluble copper and iron sulfates and insoluble basic iron sulfates. The depth of oxidation ranges from 100 to several hundred feet.

TABLE 2. - Physical properties of Kelley-mine quartz-monzonite

	Kelley-mine quartz-monzonite1/	Quarry rock from Eastern States granite2/
Apparent specific gravity:		
Average value	2.75	2.62 - 2.66
Standard deviation ..... percent	5.9	
Number of observations	5	
Compressive strength:		
Average value ..... lb. per sq. in.	16,300	30,400 - 39,900
Standard deviation ..... percent	14	
Number of observations	6	
Impact toughness:		
Average value ..... in. per sq. in.	4.70	4.9 - 7.2
Standard deviation ..... percent	16	
Number of observations	5	
Hardness, scleroscope:		
Average value ..... scleroscope units	65	88 - 98
Standard deviation ..... percent	36	
Number of observations	60	

1/ Blair, B. E., Report on Physical Properties of Quartz Monzonite From Butte, Mont., and Tactite and Hornstone From Darwin, Calif.: Bureau of Mines, Applied Physics Branch Unpublished Rept. E1-88, December 1952.

2/ Windes, S. L., Physical Properties of Mine Rock, Part 1, Bureau of Mines Rept. of Investigations 4459, 1949, 79 pp.

Ground Strength

Although the ore zones occur in the same rather limited area of intensely altered and fractured quartz-monzonite, the ground strength shows considerable variation between the zones and, with depth, in the same zone. Weathering, hydrothermal alteration, and fracturing cause weak ground, referred to by miners as heavy ground.

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The presence of joint planes and faults causes sloughing and squeezing of the walls. This requires rigid support and continuous maintenance in conventional mining.

The old stopes and workings, partly filled with timber, rock fill, or recently introduced wet tailings, have disturbed the equilibrium of the surrounding areas, causing the walls and intervening pillars to cave and crush. Old workings occupy a considerable but variable portion of individual zones and blocks. Consequently, ore estimates for individual blocks have to be reduced 10 to 40 percent. It is estimated that the overall reduction for ore reserves above the 2,000 level will average about 11.4 percent.

Ore above the 2,000 level may be classified as free-caving, which accounts for the changes in the mining procedure adopted, namely: Smaller producing areas, concrete support on the grizzly level, and measures to accelerate ore drawing.

Ore - Physical Properties

In general, the ore is relatively easily drilled. Intensely altered ore is damp and sticky; when drilled tends to plug the steel. In addition, sticky, clayey ore packs in chutes, requiring blasting, costly repairs to damaged chutes, and unexpected and undesirable delays in the flow of an ore stream. A specially designed car cleaner had to be used at the dump stations for ore sticking to the bottoms of the cars.

Another characteristic of Kelley ore is the wide variation in the size of fragments in crushed ore. Although some secondary breaking is required, oversize material and large boulders from foot and hanging walls are a problem in only a few, mostly boundary, blocks.

Broken ore occupies a greater volume than it does in place. The increase in volume, expressed by the ratio of broken ore to ore in place, is an important factor in both mine and block development and in transportation. Tonnage factors of 18.00 cu. ft. of ore per ton for broken ore and 14.41 cu. ft. for ore in place are accepted for Kelley ore. The volumetric change for Kelley ore is 1.25; this compares with 1.60 for copper porphyry ore (table 3).

Capping

The capping of ore zones which is 50 to 500 feet thick, consists of barren granite, mineralized granite completely or partly leached of sulfides, alluvium, and thin overburden. In general, the capping caves, flows, and packs as well or better than the ore does. Capping is low in grade. It may differ from ore by a reddish color, but often the changes are gradual, and the commercial ore must be determined by sampling.

When ore is drawn from a new block the rate of consolidation of capping in adjoining blocks is one of the controlling factors in preventing ore dilution.

Ore Reserves

Ore reserves totaling 160,000,000 tons containing 20 pounds of recoverable copper per ton, with minor amounts of silver and gold, are estimated above the 3,400 level.16/ Over 30,000,000 tons of the above reserves has been added since initiation of the project.

16/ Anaconda Copper Mining Co., Annual reports for 1951-54.

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TABLE 3. - Ratio of ore broken to ore in place, Kelley mine compared with Inspiration mine

Kelley mine	Specific gravity	Weight, lb. per cu. ft.	Tonnage factor, cu. ft. per short ton
Quartz-monzonite	2.72 - 2.81	170 - 175	11.77 - 11.56
Quartz-monzonite altered	2.40	150	13.33
Tailings for filling - dry	1.60	100	20.00
Used for reserves calculations:			
Undisturbed ore	2.67	166.9	12
Gob	1.28	80.0	25
Zone 9 (lower part)			12.0
Zones 1, 6, 9 (upper part)			14.0
Zone 8			15.0
Low-grade ore in place, average (estimated)			14.41
Low-grade ore - broken			18.00
Inspiration mine:			
Ore in place - average (estimated)	2.56	160.0	12.5
Ore broken - average	1.60	100.0	20.0
Ratio of ore broken to ore in place:			
Kelley ore - 1.25			
Inspiration ore - 1.60			

Above the 2,000 level the reserves are about 56,000,000 tons. About 15 percent of the copper content is in the form of oxidized minerals. At 15,000 tons daily production, equivalent to the recovery of 45,000 tons of copper annually, the estimated tonnage is enough for at least 12 years.

PREPARATORY WORK

General

Problems incident to the preparation for block caving were numerous and complex. Mine planning, designing, and programing were solved without interrupting output from other mines and while using the same facilities. Many factors that influenced certain problems were interlacing, interdependent, and even conflicting; these were carefully evaluated with regard to comparative cost, and the best compromise was accepted. Problems relative to subsidence required teamwork and the foresight of the legal, geologic, and engineering staff of the company for a satisfactory solution.

Exploration

Geologic information and numerous analyses from mining operations in the central copper zone were carefully compiled, analyzed, and summarized. An exploration program that included reopening old workings (103,700 feet), raising (2,900 feet), crosscutting (26,600 feet), and extensive sampling on nine levels, followed.[17/]

Underground exploration was confined to the mineralized area. It supplied information on the location of individual zones, their shape, size, and value. The exploration also supplied material for metallurgical tests and information for subsequent mining.

17/ Huttl, J. B., The Greater Butte Project: Eng. and Min. Jour., vol. 152, October 1951, pp. 72-81.

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The workings were sampled by three methods: Chip-channel sampling of ore in place, grab-train sampling (fractional shoveling) of broken ore, and bulk sampling. All material excavated for certain footages was shipped to the Washoe sampler, an Anaconda custom mill, for extensive tests.

Underground exploration was costly and required a great deal of time compared with the accepted practice of churn drilling used for copper-porphyry deposits.

Filling

Wet filling of old workings was first introduced at Butte in 1917 to extinguish underground fires and eliminate fire and gas hazards.[18/] Considerable engineering information had been collected from placing several million tons of filling before the block-caving project was considered. This past experience was of enormous value, because the new program required about 9,000,000 tons of filling.

Preparatory to block caving a pumping plant for filling operations was installed near the St. Lawrence shaft in 1947. First most of the workings above the 3,400 level of the St. Lawrence, Parrot, Neversweat, Anaconda, Mountain View, Pennsylvania, and Rarus mines were filled. This required 2,400,000 tons of filling and was completed before 1951. The next phase of the program, estimated to require 4,000,000 tons of tailings, includes filling the workings of the Tramway, Leonard, and West Colusa mines. Over 1,250,000 tons has been placed through the Tramway shaft already.

Although the tailings must be mined, shipped, and processed along with the low-grade ore, wet filling greatly improved mining conditions. In addition to extinguishing possible underground dormant fires and eliminating potential fire and gas hazards, the fill consolidates old coarse backfill, stabilizes the ground, cools the working places, makes development and maintenance easier, and permits better control of ventilation.

The fill material, composed of old mill tailings, had to meet rigid specifications regarding size and composition. It was excavated near Anaconda, loaded into 50-ton ore cars, and delivered by railroad to the pumping plants.

The tailings were mixed with water to obtain material of the right density and pumped under pressure at the rate of 3,000 to 6,000 tons per day through 8-inch pipes on the surface and through 6- and 4-inch pipes underground. A total of 23,000 feet of 6-inch and 140,770 feet of 4-inch pipe had been used for this operation before April 1953.

Details of filling are outside the scope of this report.

Experimental Mining

The trial blocks were in zone 6. Block caving was carried on from the St. Lawrence 600 haulage level through the Mountain View shaft. An all-gravity system (grizzly drift and branch raises) similar to that employed in the Inspiration mine was used in caving blocks 1 and 3 (fig. 4). A slusher-gravity system (slusher-drift with vertical feeder raises) was used for block 2. Available technical data are summarized in table 4.

18/ Rahilly, H. J., Mine Fires and Hydraulic Filling: Trans. AIME, vol. 58, 1923, pp. 61-76.

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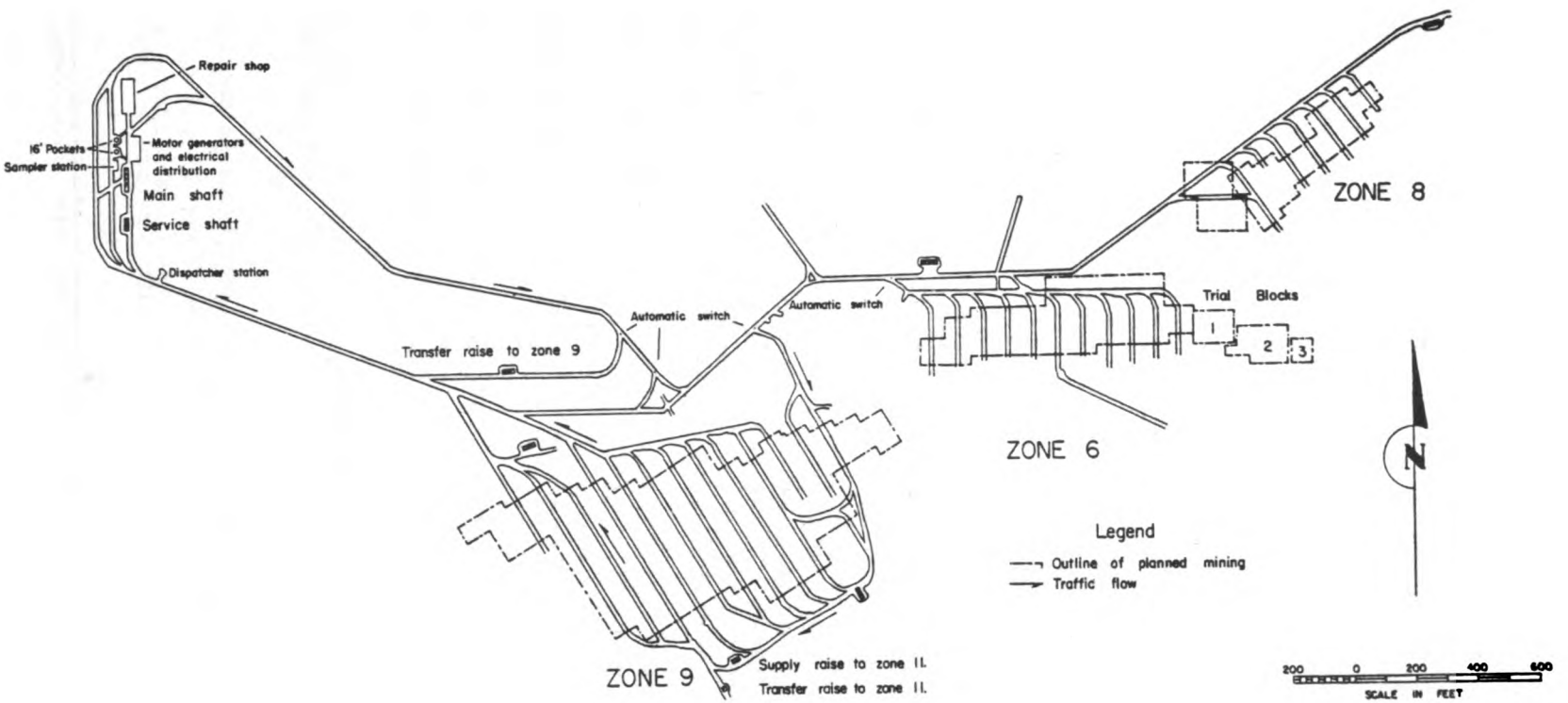

Figure 4. - Plan of 600 haulage level, Kelley mine.

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TABLE 4. - Experimental block caving - pertinent data (1945-46)

	Test block 1	Test block 2	Test block 3
Location	Zone 6	Zone 6	Zone 6
Level	600	600	600
Height of ore ..... feet	194	225	294
Size in horizontal plane ..... do.	100 x 136	About 120 x 160	68 x 75
Area ..... square feet	13,600	19,290	5,100
Method used	Grizzly drifts and branch raises; 2 haulage crosscuts.	1/Slusher-drift and vertical raises; 1 haulage crosscut.	Same as block 1; 1 haulage crosscut.
Distance:			
Floor of haulage to floor grizzly level ..... feet	52.5	36.0	
Floor undercut level to floor grizzly level ..... do.	18.0	18.0	
Number of finger raises	64	64	24
Undercut stope area served by 1 finger ..... square feet	212.5	300	212.5
Tonnage estimated	184,336	299,035	113,082
Total tonnage shipped	190,996	299,918	41,695

1/ Slusher drifts on the first floor running the entire length of the block.

Accurate records of tonnage drawn, sampling, cost of development, drawing, and maintenance were kept. Experiments were carried out to find the best grizzly spacing and the best spacing and arrangement of drawpoints. Mining over 500,000 tons of ore from these blocks provided information on the caving characteristics of the ore, tonnage extraction, metal extraction, and detailed cost of operations. The following conclusions[19] were incorporated in the design of mining in the upper levels:

(a) Advance filling of old workings facilitates mining operations and reduces development and maintenance costs.

(b) The ore caves readily, and block development does not usually require boundary weakening.

(c) The ore breaks up into pieces requiring little secondary breaking.

(d) Small size of blocks and speedy development, undercutting, and drawing are important factors in reducing mining costs.

(e) The all-gravity system is best suited to conditions on the upper levels; the excessive maintenance cost of timbered slusher drifts precludes the use of the slusher system with timber support.

(f) Drawpoint spacing with centers 20 by 15 feet is efficient for controlling of drawing and satisfactory from the standpoint of cost.

(g) An 80- by 30-foot panel is the minimum unit that can be extracted efficiently and economically.

19/ Steele, C. H., How the Greater Butte Project Was Born: Min. Cong. Jour., March 1949, pp. 32-34, 39.

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(h) Best suited to carry the ore stream from the block to the shaft without delays are: Grizzly spacing - 12 inches, width of chutes - 36 inches, and ore cars - 5 tons.

Accumulation of production experience and continued experimental work have since produced a series of changes and modifications in the original practice and design. The successful use of concrete for supporting grizzly drifts and fingers led to further experimental mining by the slusher system. A block developed in zone 8 with scraper drifts and grizzly lines concreted throughout has shown increased efficiency and lower production cost (see also Slusher-Gravity System, page 64).

UNDERGROUND FACILITIES

Important underground facilities for ore handling are illustrated in figure 5.

On the 600 level ore is hauled by 10-ton, trolley-type locomotives pulling twenty 115-cu. ft. Granby side-dump cars (fig. 6). A detailed description of the underground transportation equipment and track is given in the appendix. The train carries an average ore load of 90 to 100 tons over a distance varying from 2,000 to 4,500 feet. For rapid and simultaneous loading of cars the spacing of chutes in the all-gravity system and the spacing of slusher drifts in the slusher-gravity system is designed in an exact multiple of the car length. The rolling stock on this level consists of 11 locomotives and 225 cars. The train frequency passing the dumping station is about 7-1/4 minutes.

Dumping equipment consists of two retractable, steel dump blocks or ramps built along the track. Cars are side-dumped without uncoupling by a roller attached to one side of each car, which rides up and over the ramp with the train traveling at a fairly low speed. One storage bin is equipped with a pneumatic fork cleaner unit to loosen sticky ore from the cars in dumping position (figs. 5 and 7). The new Granby cars designed by the company have rounded corners for better discharge of sticky ores.

The two 500-ton ore pockets are cylindrical, 14 feet in diameter inside the concrete, and 65 feet in height. They are cut into the rock, concrete-lined, reinforced with 60-pound rails, and equipped with dust collectors. A third pocket and alternate haulage connection with dumping facilities are under development to double the underground-storage capacity.

Four 225-hp. motor-generator sets are installed on the 600 level to supply direct current for trolley locomotives.

A small machine shop with repair tools, welding, greasing, miscellaneous appliances, and a supply of spare parts is maintained on this level for systematic, compulsory, preventive servicing of rolling stock and emergency repairing. A special bending machine mounted on a chassis and electrically operated from a trolley line is used to bend rail and pipe.

Six-ton battery locomotives and 57-cu. ft., Granby, side-dump cars serve auxiliary levels. Each train carries an average load of 30 to 35 tons. Details of underground transportation are given in the appendix.

Ore-dumping stations, dust collectors, battery-charging stations, and car-repair stations are also maintained on auxiliary levels.

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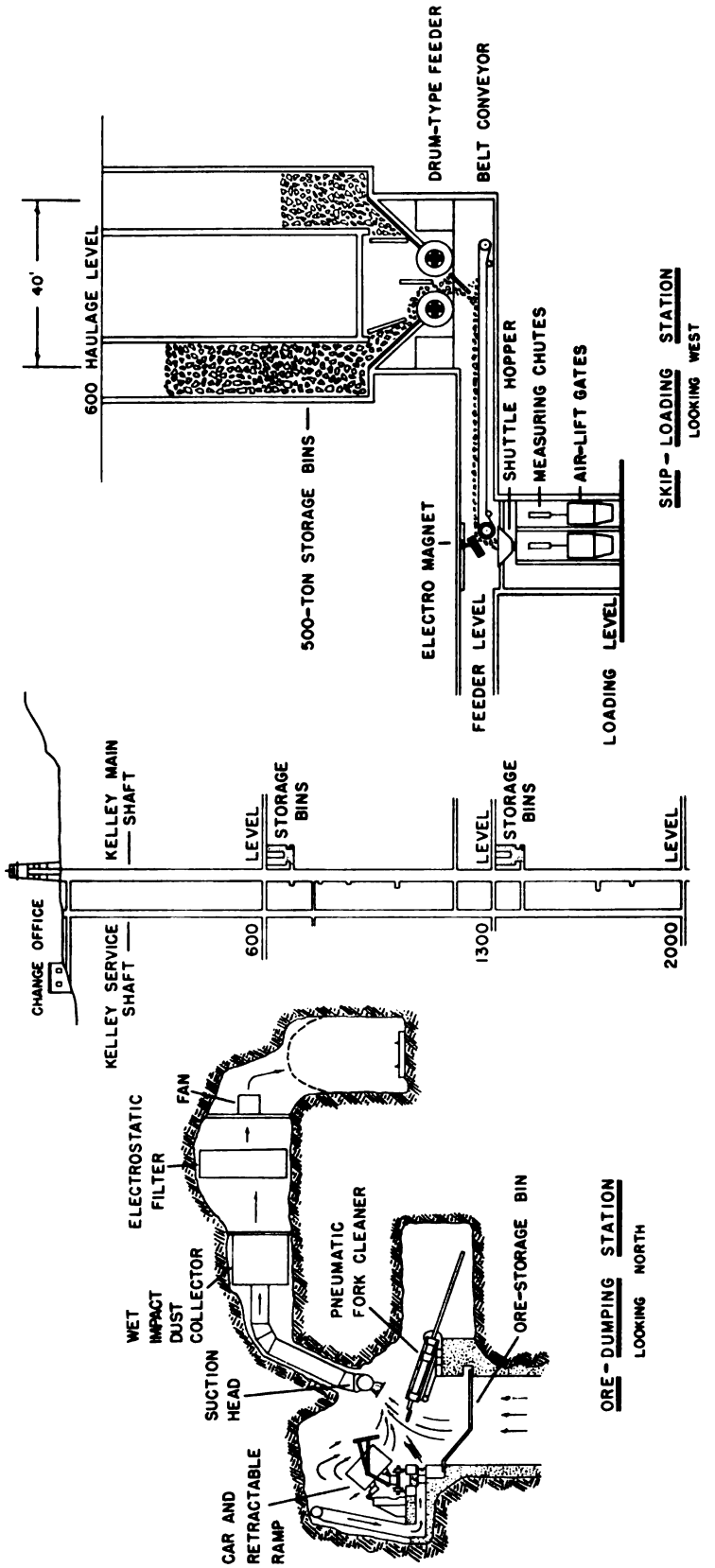

Figure 5. - Underground facilities for ore handling, Kelley mine.

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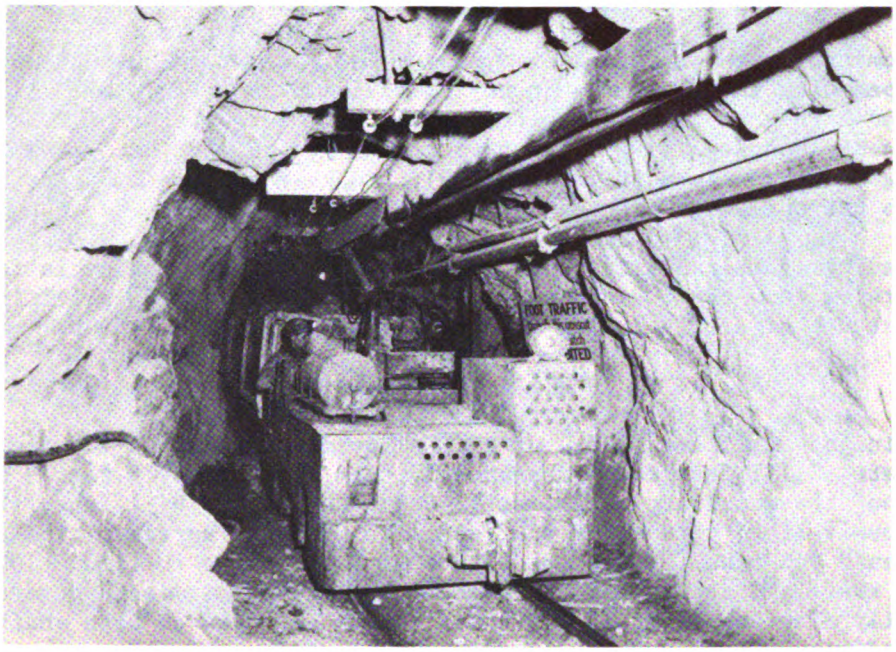

Figure 6. - Ore locomotive with loaded train of cars.

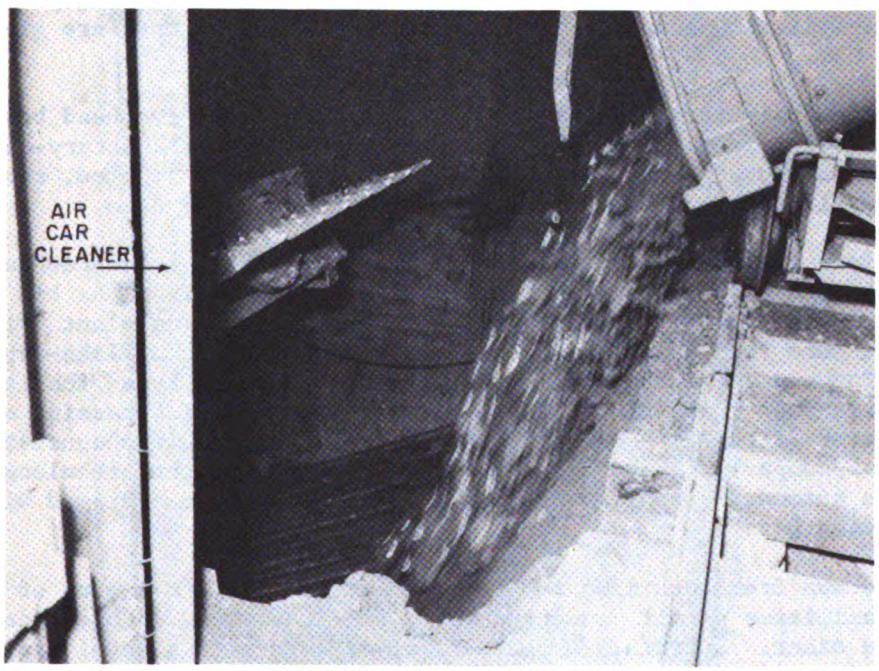

Figure 7. - Car dumping into ore pocket.

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HANDLING ORE UNDERGROUND

Ore from the pocket is drawn by a drum-type feeder to a 60-inch-wide belt conveyor equipped with an electromagnet to remove tramp iron (fig. 8). Timber is removed by hand. The ore is discharged from the belt into either of 2 measuring chutes by an air-operated deflecting hopper and then emptied into 14-1/2-ton skips for hoisting to the surface (figs. 5 and 9). To keep the ore flow in the plant in balance with the speed of hoisting, each unit is equipped with speed controls.

The loading cycle requires about 50 seconds. The hoisting cycle for each of 2 skips in service from the 600 haulage level is about 55 seconds per trip, a total vertical distance of 954 feet from loading station to dumping point.

SURFACE INSTALLATIONS

The main and auxiliary shafts of the Kelley mine are about 140 feet apart. The main shaft (2,731 feet deep) measures 35 feet 2 inches by 6 feet 9 inches in cross section; the auxiliary shaft, 2,162 feet deep, is 18 by 7 feet. Both shafts are continuously lined with reinforced concrete from the surface. The arrangement of compartments and other details of both shafts are shown in figure 10. The size of the main shaft and the capacity of the hoisting machinery were governed by the proposed daily mine output of 15,000 tons. Pertinent data on both shafts are given in the appendix.

A short, concreted adit connects the combination changehouse-mine office building and the mine yard with the landing stations of the main and auxiliary shafts (figs. 11 and 12).

Surface structures include a hoist building, coarse crushing plant, waste and ore-storage bins, changehouse-mine office, and a sample processing plant for drying, crushing, and splitting samples. Details of surface structures are given in the appendix.

The changehouse-mine office building is equipped with overhead basket lockers, showers, and toilet facilities, and includes a lamproom with battery-charging racks and a storage warehouse for supplies. The building is well-lighted, steam-heated, adequately ventilated, and supplied with a public address system.

The surface flowsheet is as follows. The ore hoisted is dumped automatically from the skip into a 1,400-ton-capacity circular bin; ore from the bin is fed by an apron feeder to the vibratory grizzly of the primary crushing plant (fig. 13). The oversize ore, more than 6 inches in diameter, is crushed in a Blake-type jaw crusher (fig. 14). The undersize and the crushed ore are elevated by an inclined conveyor belt to the transfer tower at the top of the ore-storage bin, passing en route over a weighing meter (fig. 15). The 7,500-ton-capacity storage bin consists of 2 rows of 16 circular compartments; a cross-conveyor distributes the ore along the top of the bin by an automatic tripper (fig. 16). The hoppers are equipped with air-operated loading gates.

Ore is drawn from both sides of the ore bin into railroad cars of about 55-ton capacity. Facilities permit loading a total of 21 cars, 10 on one side of the bin and 11 on the other. A trainload consists of 42 cars with a total capacity of about 2,300 tons. A special branch of the company's own Butte, Anaconda & Pacific Railway connects the mine with concentrating and smelting plants at Anaconda, Mont.

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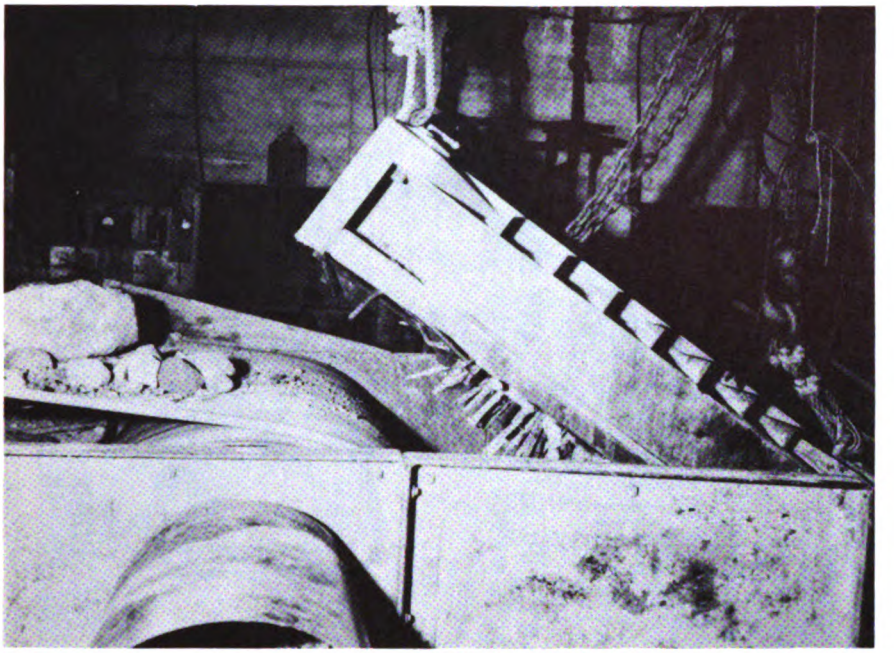

Figure 8. - Belt magnet.

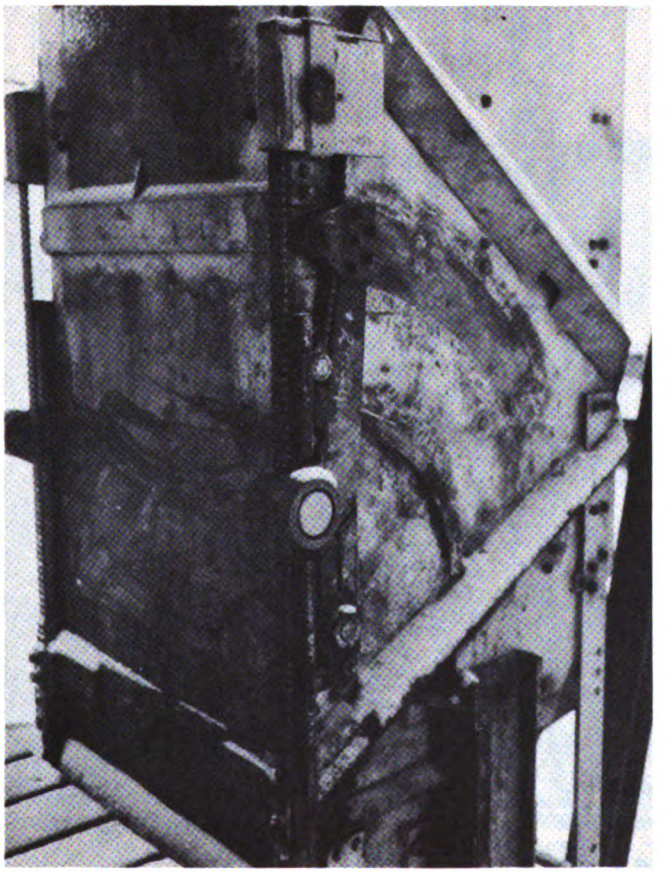

Figure 9. - Bottom dump skip, showing cam arm dumping mechanism.

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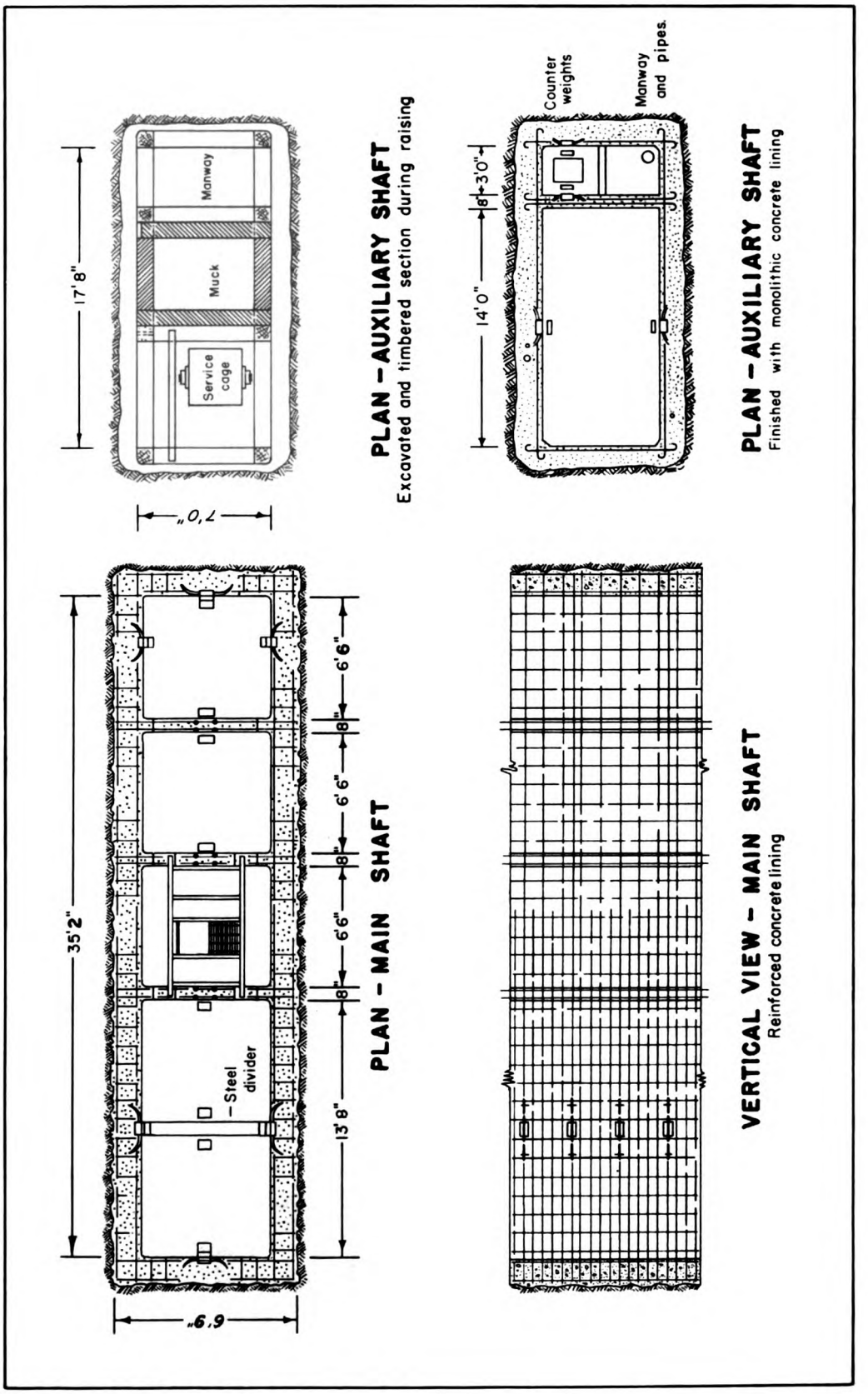

Figure 10. - Main and auxiliary shafts, Kelley mine.

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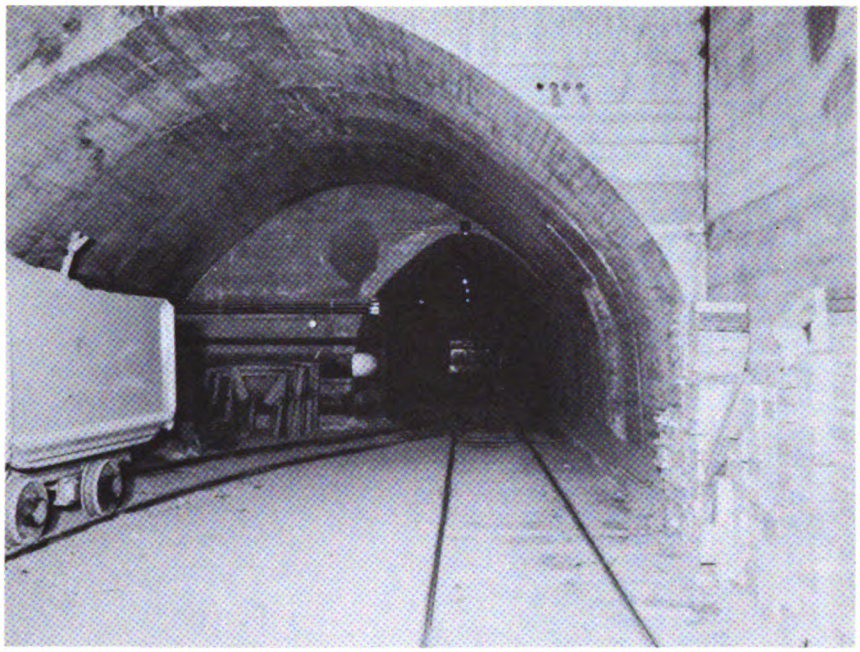

Figure 11. - Inside Kelley adit—looking toward main shaft station at 35 foot level.

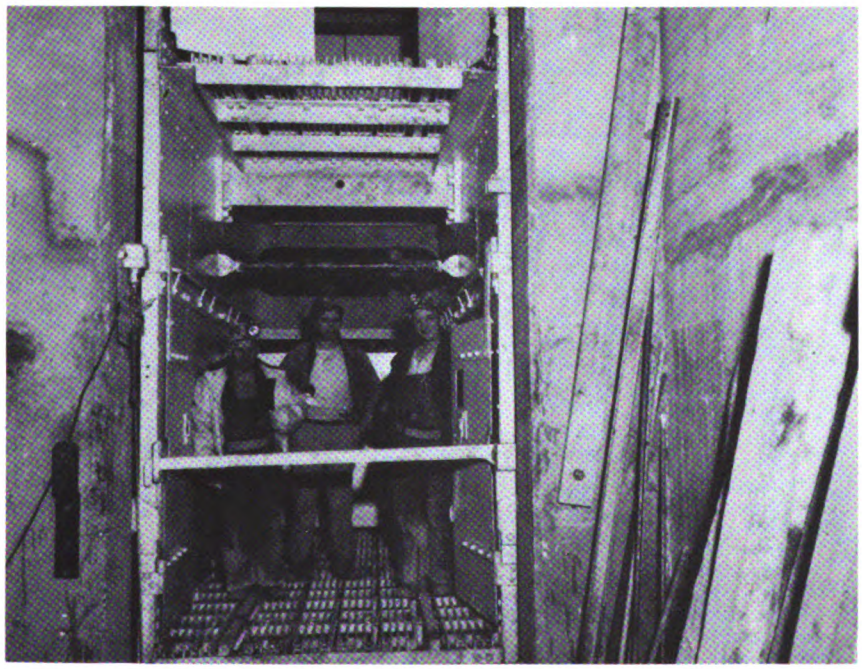

Figure 12. - Double-deck service cage.

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Figure 13. - Crushing plant–drum feeder and vibrating screen.

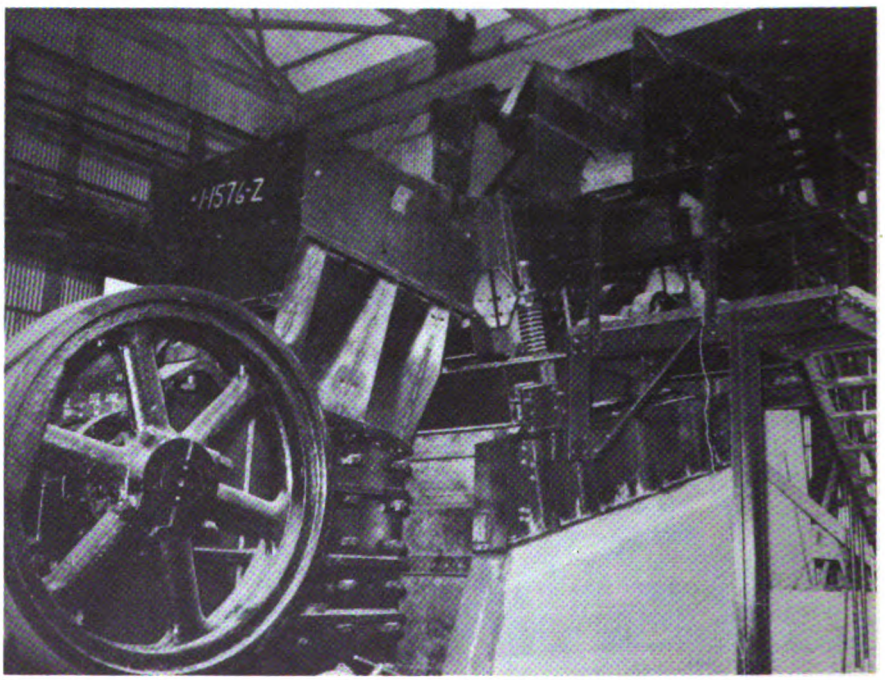

Figure 14. - Crushing plant–jaw crusher, 100 hp.

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Figure 15. - Conveyor belt moving ore from crushing plant to ore bins.

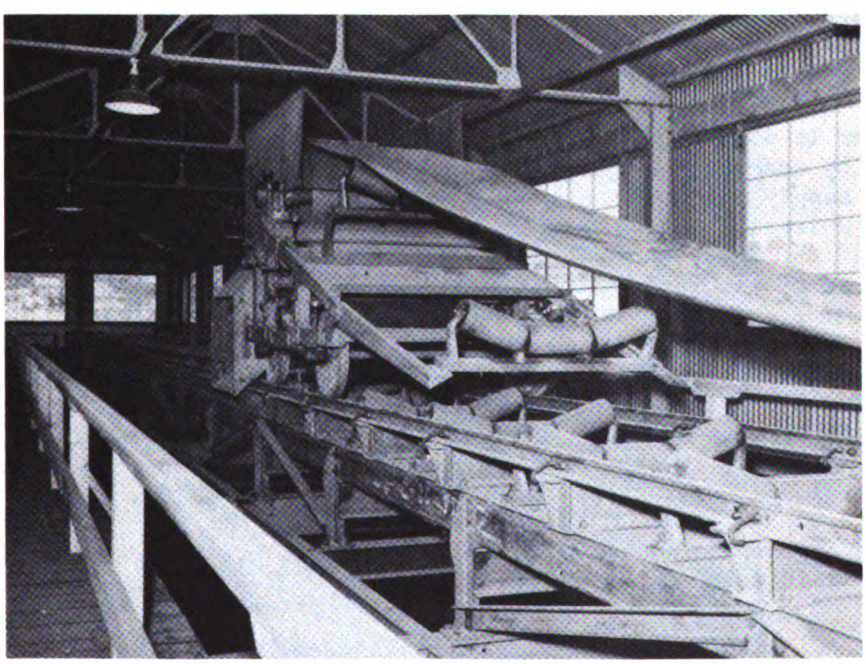

Figure 16. - Distributor and distributor belt at the top of ore bins.

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At the plant the ore is weighed in railroad cars and after unloading and crushing is mechanically sampled. Every week the smelter and mine data are compared to determine correction factors for car weight, skip weight, and mine grade.

SUPPLIES AND MATERIALS HANDLING

The Kelley mine is an integral part of the Butte operations whose power, compressed air, and other facilities are centralized to serve all mines.

Power in the Kelley mine is completely electric. Electric power is supplied by the Montana Power Co. from several hydroelectric plants on the Missouri River and its tributaries. The 3-phase, 60-cycle, high-volt (50,000) current is stepped down to 2,400 volts at a substation near the mine. The hoists are operated on 600 volts d.c.; the smaller motors and lighting are operated on 440, 220, and 110 volt a.c.

Compressed air for mining equipment is supplied through a pipeline system from a central compression plant near the High Ore mine, one-half mile from the Kelley mine.

Steam is provided by central heating plant, which is operated with natural gas supplied by the Montana Power Co., and is near the compressed-air plant. The steam, supplied by pipelines, is used for heating the office building and for clothes drying.

Water is furnished by the Butte Water Co., which provides an ample supply from its Big Hole River plant about 25 miles southeast of Butte. A water reservoir to supply the mines is on the hill, and a system of surface and underground pipes furnishes clean water for the Kelley-mine operations.

Timber for the mine is delivered for the most part in railroad cars to the Kelley-mine yard direct from the company mill at Bonner, near Missoula, Mont. The remainder of the timber is brought to the mine yard by truck from the central framing plant and from the central timber yard.

The timber is precut, packaged, and stored on skids for easy handling (fig. 17). When required by underground operations, the timber is loaded by a 5-ton forklift truck on mine supply cars, hauled to the shaft, and lowered and delivered to any desired haulage level, zone, and block.

Dynamite is stored in specially excavated chambers on each level.

Concrete is supplied by the central concrete-mixing plant 1,800 feet northeast of the Kelley mine. The plant, of about 40-cu. yd.-per-hour capacity, has an automatically controlled weight batcher and is equipped for year-round operations. The ready-mixed concrete, loaded into 2- or 3-cu. yd. transit mixers, is delivered to the mouth of the shaft, or churn-drill holes, and then transported through pipelines underground as close to the place of use as possible (fig. 18).

Track and pipe and other material used in mining are standardized and stored in the mine yard or in warehouses.

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Figure 17. - Packaged crib raise timber stored on skids in mine yard.

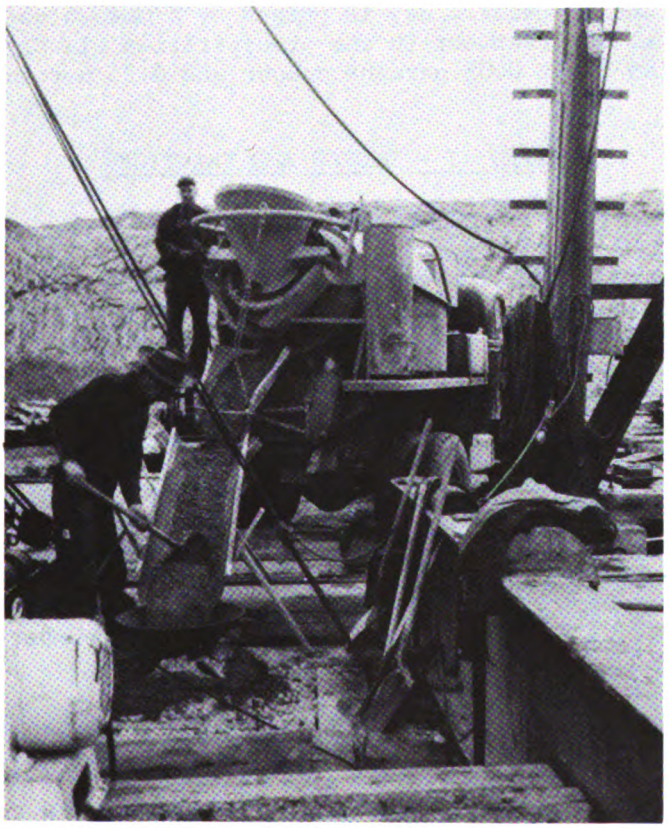

Figure 18. - Batch concrete truck pouring concrete into churn-drill hole for underground use.

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COMMUNICATIONS

Direct telephone lines connect the mine office building with the company switchboard and with the Butte public telephone exchange. Special lines connect the office with underground stations on each level and with auxiliary services plants. A communication system installed underground permits direct conversation between interdependent operations, such as the slusher and the grizzly levels of the same block or the concreting crew in any zone and the ready-mix delivery point on the surface.

On the 600 level haulage is directed from a central dispatch station near the ore-storage bins by two-way radio, the trolley wire serving as the source of power and carrier medium of the waves (fig. 19).

A dispatcher regulates the movements of all trains by direct contact with the operators and charts their movement with pegs on the map.

DRAINAGE

Underground waters of the Kelley-mine workings are drained by gravity to the High Ore shaft through levels common to all mines. Two parallel diamond-drill holes drilled from the 600 level to drainage workings on the 1,100 level form a part of the Kelley-mine drainage system.

About 30,000 tons of water a day is pumped by a modern underground pumping plant to the surface for treatment in the precipitation plant. The water is highly corrosive and carries about 0.03 percent copper and 0.75 percent solids in suspension.

PART II - DESIGN AND PRACTICE

SCOPE AND GENERAL FEATURES

The initial stage of the Greater Butte project includes developing 56,000,000 tons of copper-ore reserves above the 2,000 level and mining and hoisting 15,000 tons of ore daily, equivalent to an annual output of 4 million tons of ore or 40,000 tons of copper in terms of recoverable metal. This output is in addition to the company annual production of about 50,000 tons of copper from other mines in the Butte district.

To assure steady output by block caving, the ore zones are divided into self-contained units that can be opened, developed, and mined gradually, independently, and according to schedule. The zones are divided by levels into lifts in the vertical plane and subdivided by crosscuts into blocks in the horizontal plane. Because each block is capable of yielding only a limited output, several blocks have to be drawn to make up the total daily output; consequently, a number of blocks have to be prepared ahead of extraction.

In general, the scheduled output from the Kelley mine calls for 2 production levels, 1 level under development, and 1 new level being opened by deepening a shaft.

The principal mine-development workings include the main Kelley shaft to the 3,000 level, the auxiliary Kelley shaft to the 2,000 level, main haulageways on the 600, 1,300, 1,600, and 2,000 levels leading to ore zones, intermediate levels for development and ore drawing of individual zones, transfer and service raises, and

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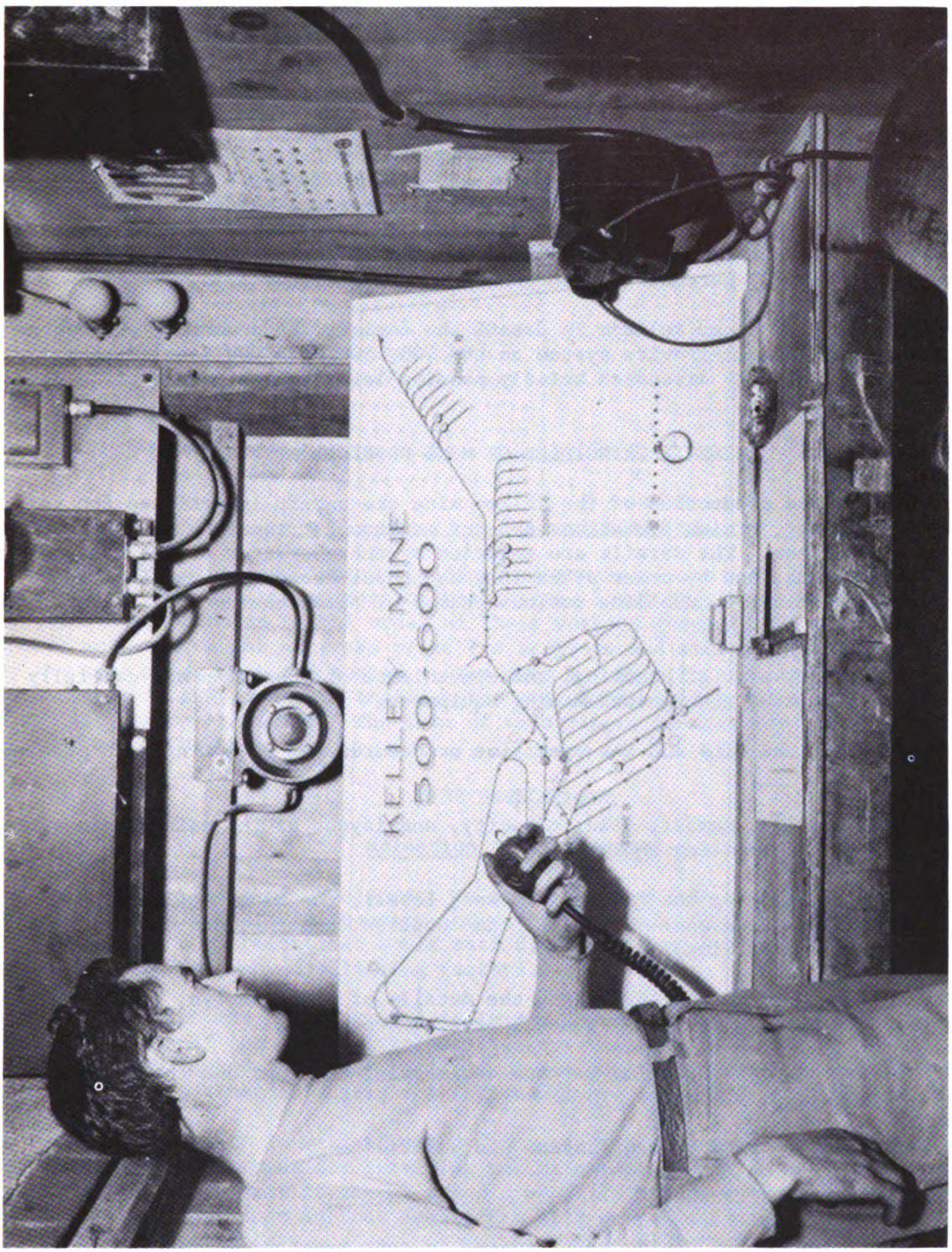

Figure 19. - Dispatcher's station on 600 level.

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special workings for ventilation and other purposes. These workings are connected with several old shafts in the area for ventilation, drainage, and safety (emergency exits).

At present ore drawing is confined to zones 6, 8, 9, and 11 above the 600 level and zone 1 above the 1,300 level.

Originally all ore was extracted by the all-gravity system. As a result of the introduction of concrete support of development workings and subsequent successful mining experiments, an increasing number of blocks is being mined by the slusher-gravity system. Output by the latter system reached 15 percent of mine production in November 1954 and 46 percent of mine production in November 1955.

This part of the paper relates in detail the design, the practice, and operational changes of the all-gravity system on the 400, 500, and 600 levels; the slusher-gravity system is discussed briefly pending later special study of lower levels.

GENERAL PRINCIPLES OF MINE DEVELOPMENT

Development and production of the Kelley mine are carefully designed for safe, continued, efficient, mining operations and for maximum ore recovery at the lowest possible cost per ton. The details are governed by the character of the ore bodies, by ground conditions, and by other often interdependent factors of varying importance. Geologic factors controlling various phases of block caving are illustrated in table 5.

Head clearance, ease of handling, and convenience of operation are carefully considered in selecting underground mining equipment.

In laying out the mine and the operative procedures the following main problems were studied:

(a) The location, number, size, capacity, and layout of the main openings; the size and capacity of hoisting equipment.

(b) The selection of the interval between levels; the amount of vertical and lateral development in advance of mining; the location, spacing, and direction of the main development workings.

(c) The system of block caving and the details of block design, such as size, orientation, grizzly spacing, and sequence of block operations.

(d) The method of supporting various development workings and the resultant width of the openings.

(e) The requirements of ventilation and safety.

(f) Surface subsidence.

(g) The economics of mining as expressed in cost per ton of ore.

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TABLE 5. - Geologic factors controlling various phases of block caving

	Size, shape, and attitude of ore bodies	Character of ore		Capping
		Physical	Grade	
1. Location and layout of main openings..	X			
2. Selection of mining horizons	X			
3. Location and layout of haulage levels.	X	X		
4. Location of ore passes	X			
5. Size of blocks, drawpoint spacing amount of undercutting and boundary weakening	X	X		X
6. Orientation of blocks	X			
7. Sequence of mining blocks and block development details	X	X		
8. Support of openings		X		
9. Dilution			X	X
10. Daily production	X		X	
11. Cost	X		X	
	X - major importance			

Plans for mine development are designed and initiated 6 to 10 years ahead of production. Wet filling, deepening the shafts, level development, and other major projects are scheduled in the order of their working sequence and interdependency. Charts and maps on the progress of active projects and mining operations are kept by the mine management and posted at the mine weekly. In addition, a periodic progress report is prepared that includes the status of all development workings as compared with those scheduled. The form of this report is discussed further in the chapter on ore drawing.

MINE DEVELOPMENT

Selection of Shaft Site

The main purpose of development is to make accessible a maximum of ore with a minimum of development; in choosing the number and the type of main development openings, the best compromise that would satisfy the economics of the mining operation was sought for the many conflicting factors.

The position and the nature of individual ore zones, the topography and the physical nature of the ground, the possibility of subsidence, and the presence of other mines indicated the practicability of one permanent, modern, deep shaft instead of several separate shafts for individual ore bodies (fig. 20).

The Kelley shaft is well-situated for installing a surface plant and to existing subsidiary facilities for servicing and maintenance; it has good access to surface transportation facilities for ore and supplies and is sufficiently remote from active mining areas that surface subsidence will not affect the surface and underground mine installations.

The cross-section dimensions and other features of shaft design were determined on the basis of required output, estimated life of the mine, modern mining practice, available equipment, and cost. Because of predetermined capacity, heavier hoisting units than were used at other properties in the district were selected.

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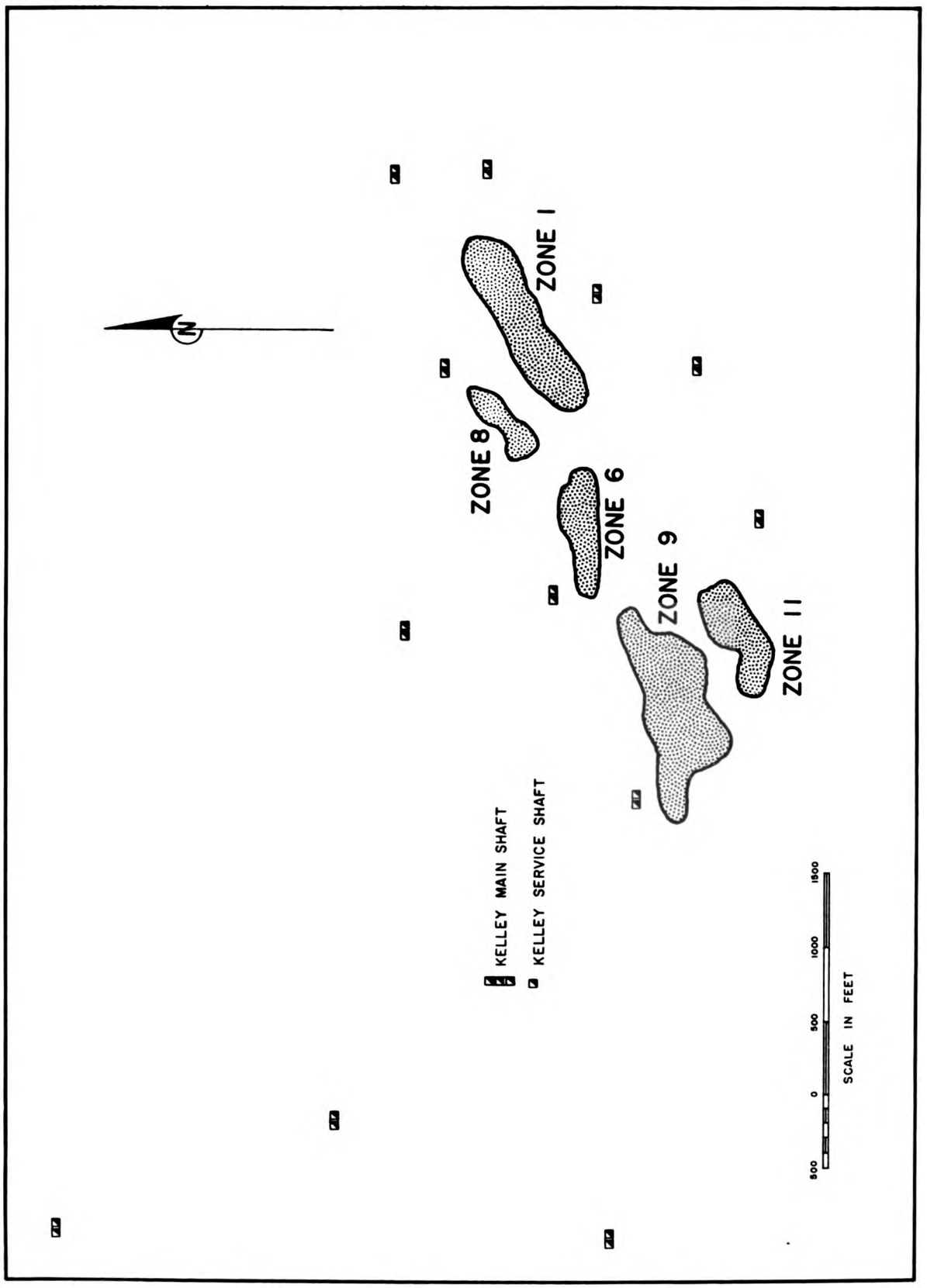

Figure 20. - Location of Kelley shaft in relation to zones.

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Selection of Main Haulage Levels

Production requirements, mining experience, and development cost govern the location and design of haulage levels. The greater the interval between levels the lower the development cost and time period required for mine development; thus, higher ore columns are developed at less cost per ton. Concentrating haulage on a few levels saves the cost of extra shaft stations and ore pockets, requires less investment for haulage equipment, and cuts the operational cost of hoisting. Centralized output aids efficient supervision and safety of all mining operations.

In addition to the size, shape, and attitude of the ore bodies the controlling factors governing the interval between levels are the speed of ore drawing and the amount of dilution by whichever system is selected. In turn, the system depends on the nature of the ore and on production requirements. Another important requirement is that mining be completed and the level abandoned within the time it can be kept open without prohibitive maintenance cost. In block caving the present trend is toward an increased height of the ore column mined from one level.

Selection of the height of ore column that best can be mined from one level and hence selection of intermediate levels form a compromise between economics and practicability. At the Kelley mine a vertical ore column 300 to 400 feet high is considered best for profitable and efficient mining. For a column over 300 feet high the development cost per ton of ore changes at a relatively low rate; on the other hand, in drawing ore from a column exceeding 400 feet the savings on development may be lost by dilution. In higher-than-average-grade ore the height of the column may be decreased proportionally to the total metal content in such column. In steeply dipping ore zones a height of less than 50 feet of average-grade ore is considered unprofitable for boundary panels in the hanging wall.

With these principles governing the interval the main haulageways were chosen at the 600, 1,300, 1,600, and 2,000 levels.

The 600 level gives access to and serves directly zones 6, 8, and 9 and, by means of the 400 and 500 intermediate levels, zones 9 (upper part) and 11. Average mining lifts for the 600 level are:

Level	Zone	Lift, feet
400	9	350
500	11	200
600	6	250
600	8	300
600	9	300

The 1,300 level gives access and provides haulage and ventilation service directly to zone 1 and, by means of the 800 and 1,100 intermediate levels, zones 6 and 8. Average mining lifts are:

Level	Zone	Lift, feet
800	6	277
1,100	6	280
1,100	8	550
1,300	1	313

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The 1,600 level will serve zone 1 and possibly zones 4 and 5, and the 2,000 level will give access to and serve zones 1, 3, 4, and 5. The general layouts of the 600, 1,300, and 2,000 haulage levels are given in figures 4, 21, and 22.

In locating mining levels special consideration was given to existing workings in the area for their possible use in the long range program of development, such as for access from other shafts; thus, the 600 level of the Kelley mine overlaps an old level of the Mountain View mine.

Main Haulageways

General (Layout and Design)

Haulageways provide access to minable zones and an outlet for ore mined. The life of these workings is determined by the amount of reserves available through each haulage level, the rate of mining, and their possible future use after the block has been mined. Although the shortest distance between the shaft, or transfer raise and the ore zone is the aim of level development, the original cost, the yearly cost of maintenance, and other factors are carefully appraised in planning the layout and design of haulageways.

In selecting the layout and design of haulageways at the Kelley mine the following factors were considered: The tonnage to be handled during one shift, the length of the haul, the requirements of safety and ventilation, and the present trend of underground transportation toward faster speed, heavier locomotives, larger capacity of ore cars, and longer trains.

Insofar as possible, the haulageways were placed in ground allowing the maximum possible footage unsupported or supported with rock bolts. They were kept clear of major faults whose strikes approximate the bearings of the workings and of old stopes that might have changed the equilibrium of the ground.

Stations are designed to permit locomotive drivers to see the entire length of the train when dumping at the main shaft or loading supplies at the auxiliary shaft.

600 Haulage Level

On the 600 level a loop haulage system with clockwise traffic is used to handle production from zones 9 and 11. It consists of 2 main laterals extending from the shaft to the mining area in the foot and hanging walls of ore zone 9, permitting direct haulage from this zone and haulage from the transfer raise connected with the 500 level of zone 11 (fig. 23). For zones 6 and 8 on the same level a branch haulage system is used (fig. 24). The movement of trains on the level is regulated by a dispatcher.

Haulageways are driven with a cross section of about 9 by 9 feet in the clear, necessitated by using 115-cu. ft., Granby, side-dump cars. Drifts are untimbered or supported with rock bolts and wire grids in sections requiring light support (figs. 25 and 26). In heavy ground 3- and 5-piece timber sets are used (fig. 27).

Track is 60-pound rail laid on 36-inch gage with a grade of 0.3 percent in favor of the load; there are standard 75- and 100-foot turnouts for main laterals. To maintain the track in good condition slag ballast and close tie spacing are used when necessary. In addition to standard spring-throw switches, several remote-controlled automatic switches are installed at main intersections.

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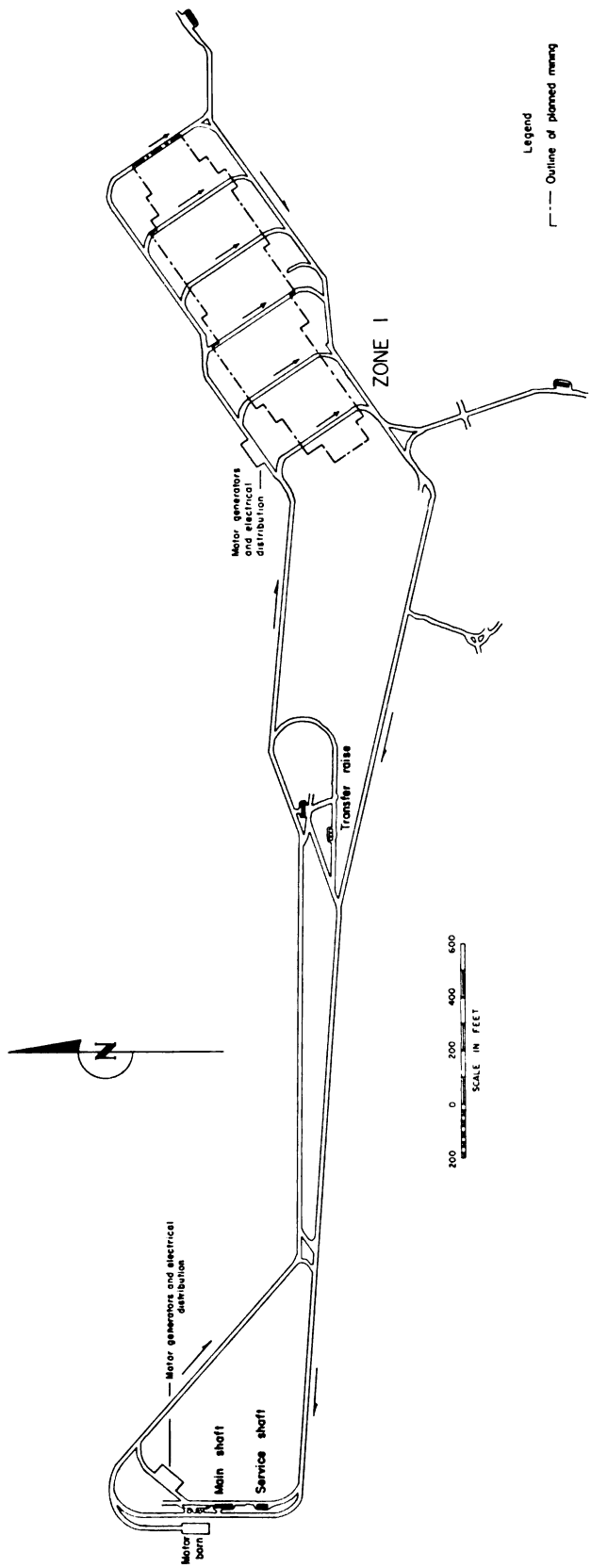

Figure 21. - Plan of 1300 haulage level, Kelley mine.

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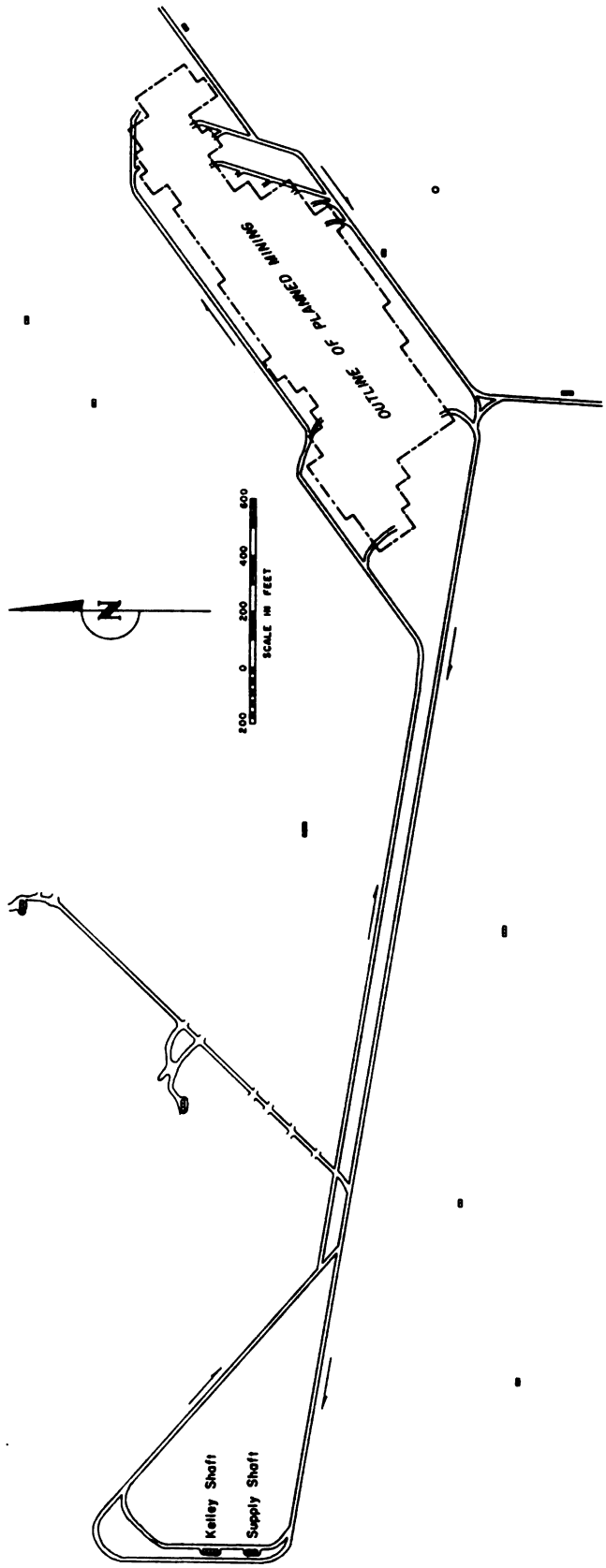

Figure 22. - Plan of 2000 haulage level, Kelley mine.

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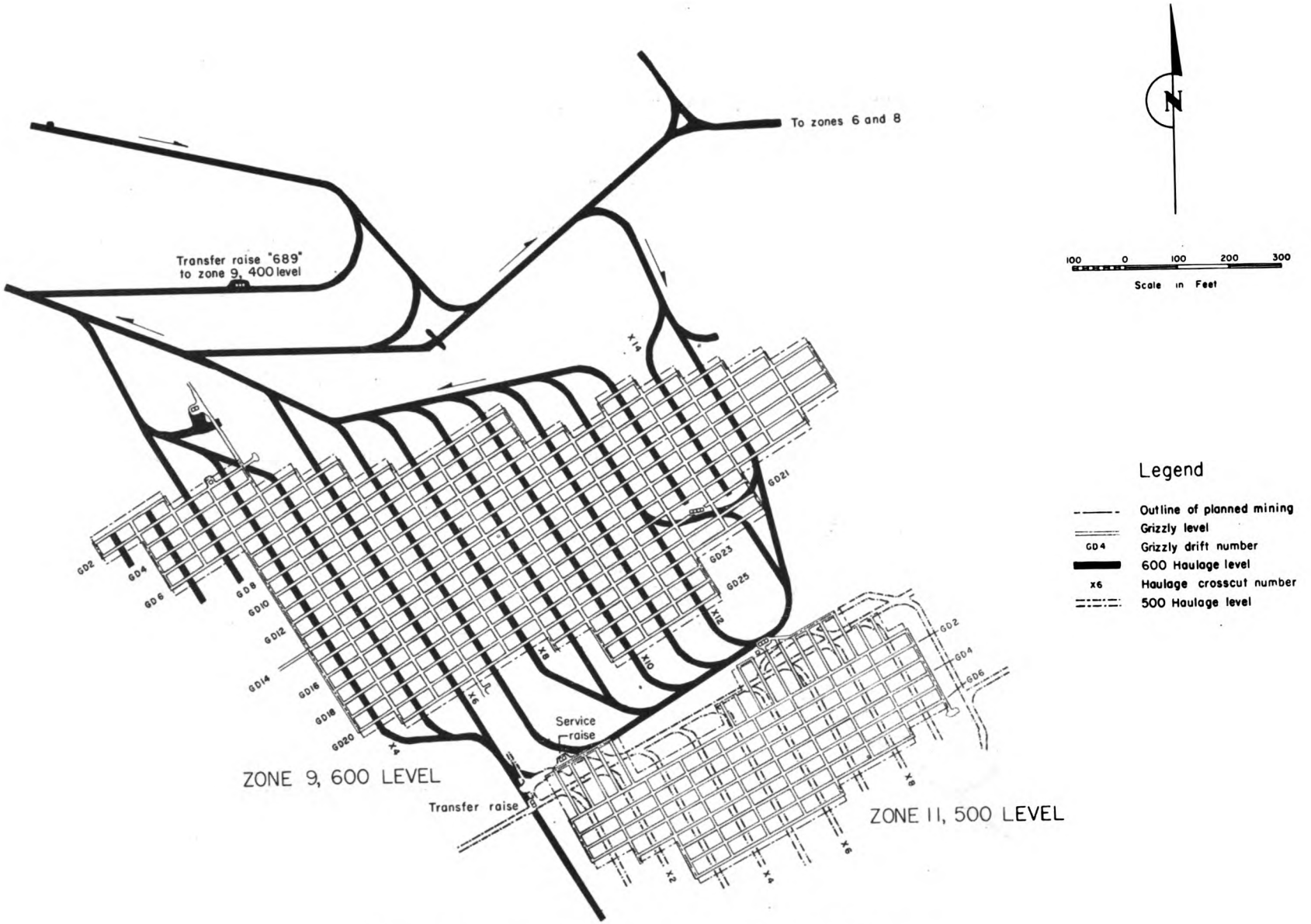

Figure 23. - Zones 9 and 11 on 500-600 levels, Kelley mine.

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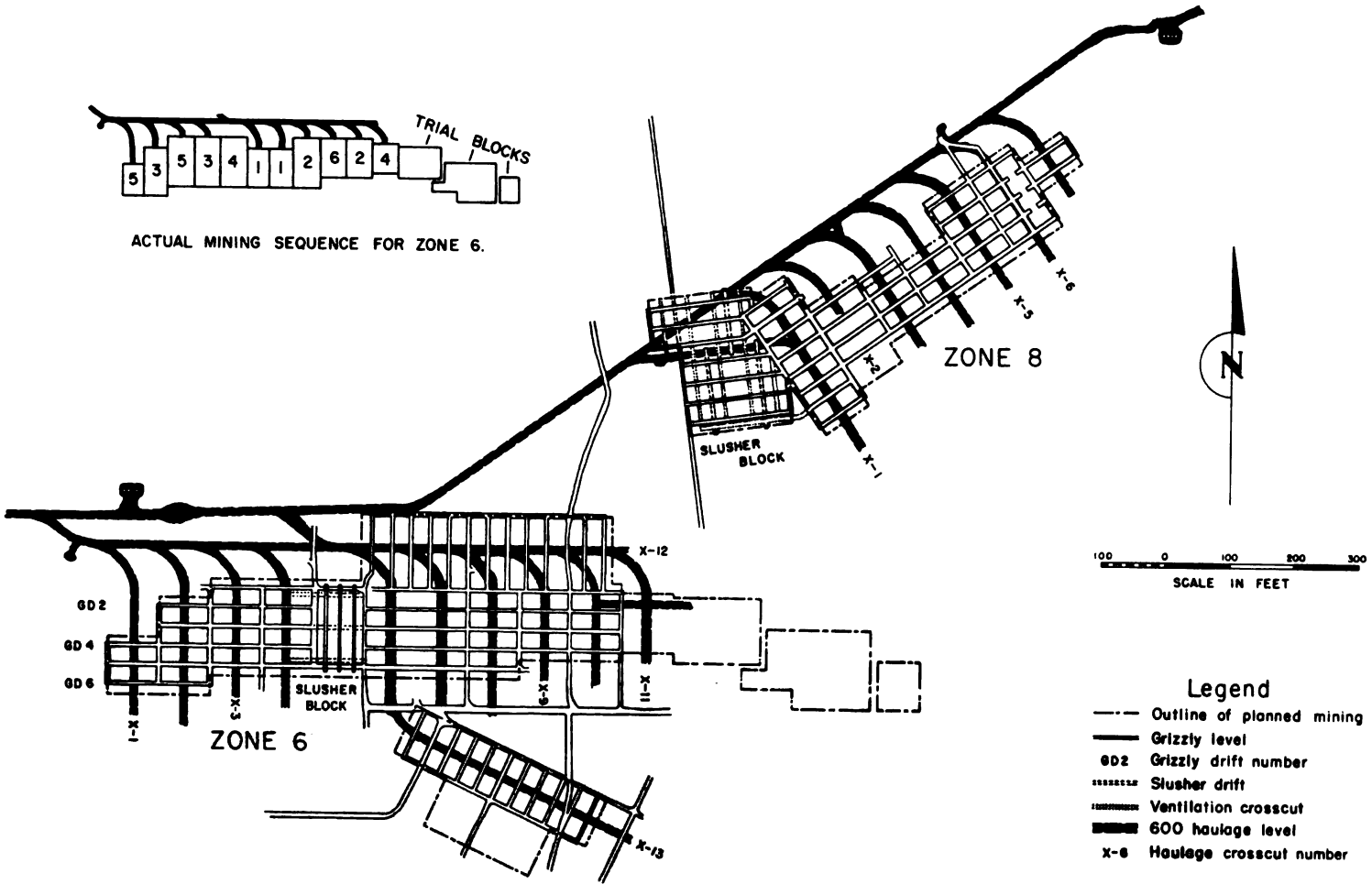

Figure 24. - Zones 6 and 8 on 600 level, Kelley mine.

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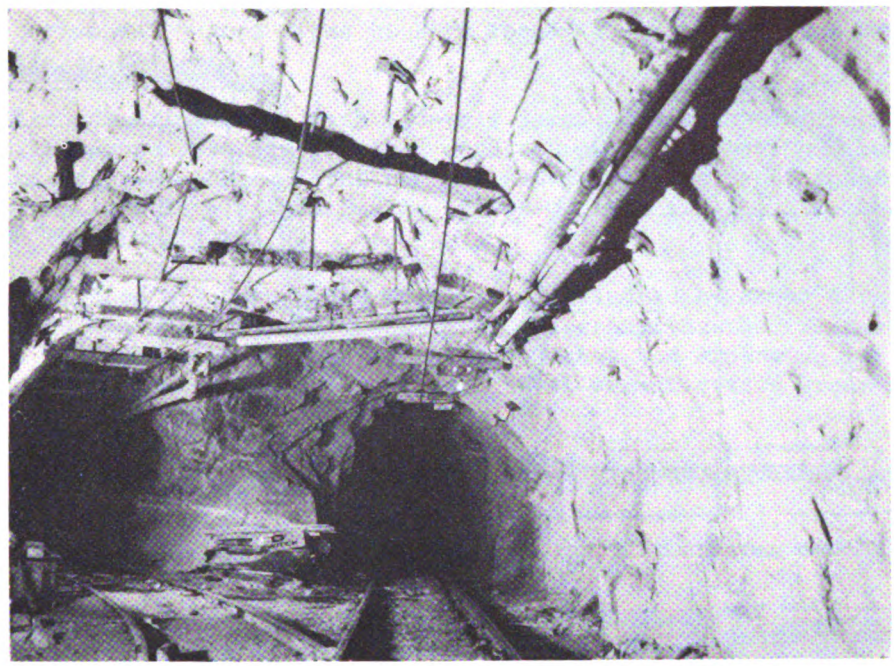

Figure 25. - Rock-bolted intersection.

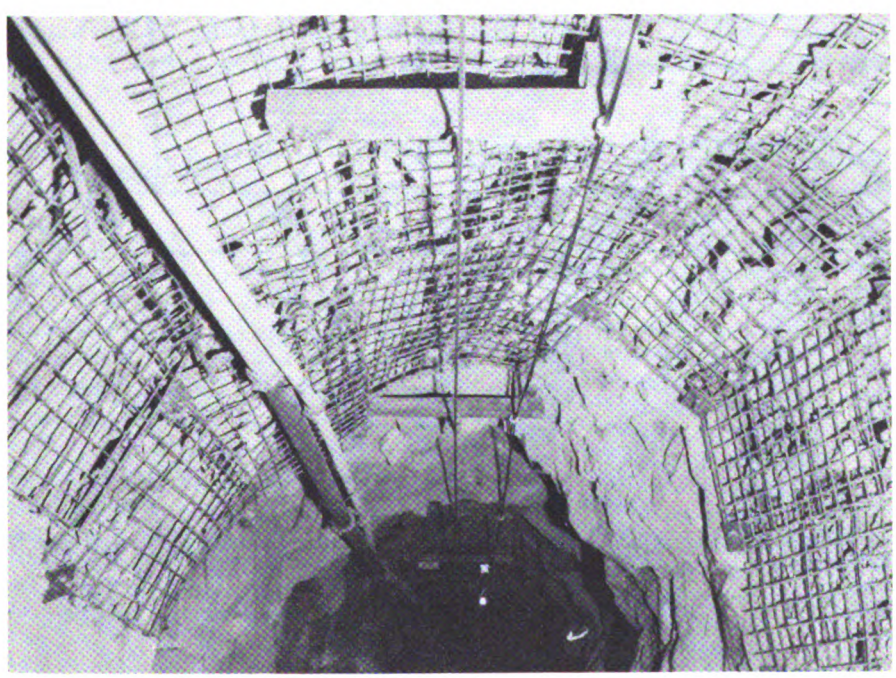

Figure 26. - Steel grids held up by plates and roof bolts.

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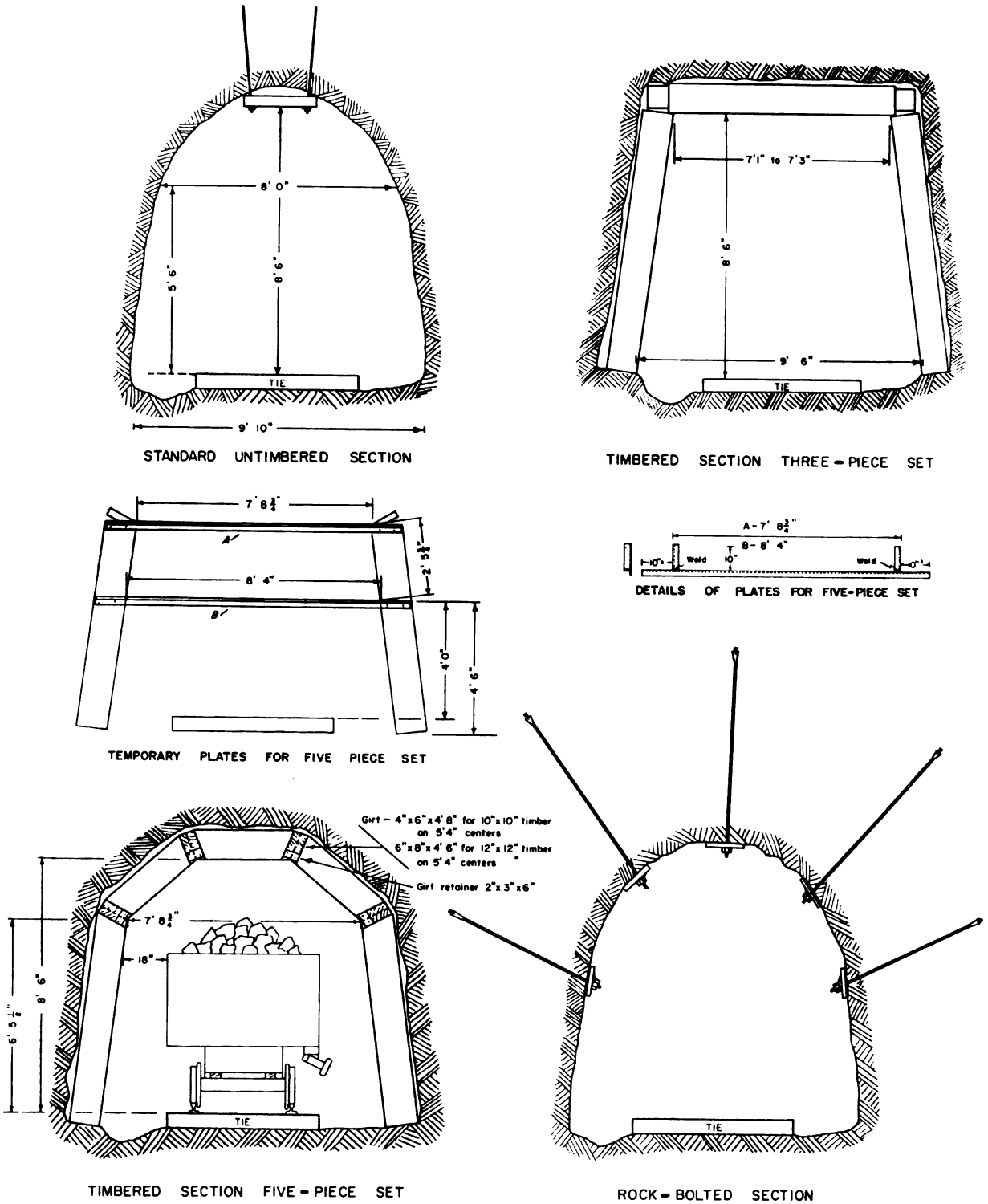

Figure 27. - Haulage level, 36-inch-gage track, standard cross section and support.

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Trolley wire, power cable, air and water lines, and telephone lines are suspended from horizontal 4-foot cross timbers spaced 15 feet apart and hung 8-1/2 feet above the rails on rock bolts.

The drifts are illuminated with 110-volt lamps. Curves, dangerous intersections, and blind approaches are provided with red warning lights and buzzers operated by the train crew. Safety stations for pedestrians at about 150-foot intervals are marked with amber lights. Automatic remote-control switches are equipped with red and green traffic lights.

The installation of underground track is facilitated by using a rail-bending machine mounted on a chassis and electrically operated from the trolley line. The track crew is equipped with pneumatic impact wrenches and explosive rail punches; it is supplied with precut and packaged wood ties, cement-coated track spikes, and other necessary materials.

A 36-inch-gage track cleaner expedites the maintenance and cleaning of haulageways.

Drifting

Main haulage drifts are driven with a specially designed boom-type jumbo drill equipped with drifters capable of drilling 8-foot holes and permitting fast set-up and removal of equipment (fig. 28). Previously, a column-and-bar-mounted drill and 5-foot round was used. According to extensive tests, the results of which are summarized in table 6, the overall drift advance with the boom-type carriage is 4.20 feet per shift, about 40 percent more than with the bar-mounted drill. This rate of advance in drifting was possible due to speedup in the mucking cycle, effected in part by development of the Anaconda-type car transfer (fig. 29).

The standard drill pattern consists of 30 holes - 5 holes, drilled in a vertical line to form a slot and 2 holes, the rakers, angled toward the slot comprise the burnt cut used.

TABLE 6. - Drifting; time study

	Boom-type drill carriage	Column bar-mounted drill
Shifts	5,274	
Men per shift	2	2
Round feet	8	5
Cross section ... sq. ft., average	1/102	96
Advance per man-shift ... feet, average	2.10	1.49
Peak advance per man-shift	3.83	
Holes per round	30	
Feet drilled per round	240	
Ratio 1 hole to area ... sq. ft.	3.40	
Cubic feet per foot of drilling	3.40	

1/ Timbered section 10-1/2 by 11-1/2 feet or 121 sq. ft., untimbered section 8-1/2 by 9-1/2 feet or 81 sq. ft.

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Figure 28. - Two-boom jumbo drill operated by one man.

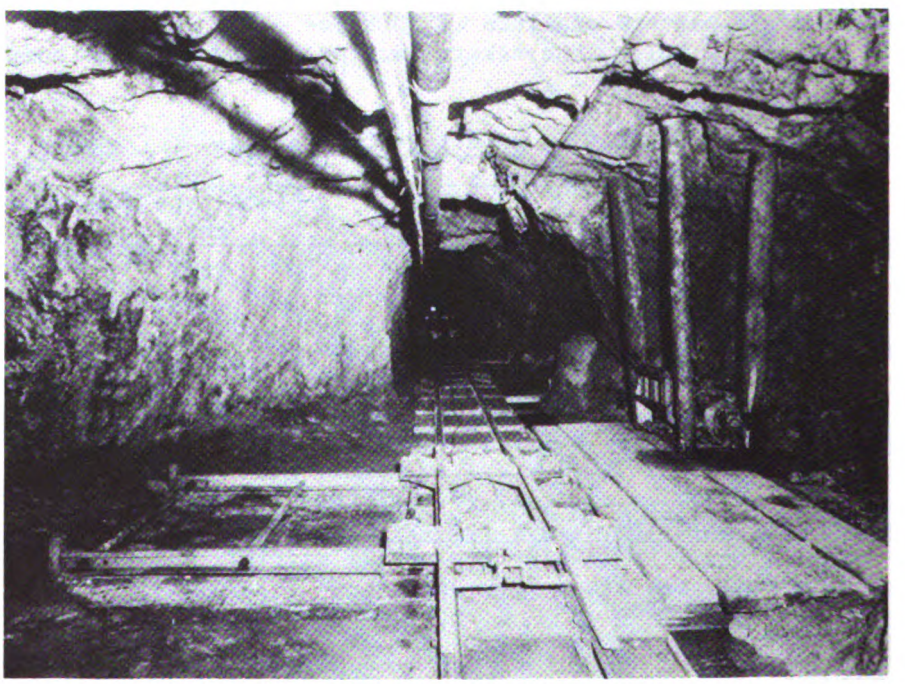

Figure 29. - Car transfer.

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Intermediate Haulage Levels

Intermediate haulage levels are connected with main haulageways by ore transfer and supply raises. The latter are equipped with air-operated gear hoists and service cages. The 400 level of zone 9 is connected by the St. Lawrence and "689" transfer raises, and the 500 level of zone 11 is connected by the "603" transfer raise with the 600-level main haulageway (fig. 30). The branching-line system for ore haulage on both intermediate levels consists of laterals driven from the dumping stations at transfer raises parallel to and in the footwall of each zone. Crosscut haulage branches from laterals are laid out across the zone 80 feet apart.

The ore-pass dumping stations are equipped with car-cleaning, air-operated, fork and dust collectors similar to those on the 600 level.

Six-ton battery locomotives and 57-cu. ft., Granby-type cars are used for haulage. The drift has a cross section of 7 feet by 7 feet 4 inches in the clear. All turnouts have a standard 40-foot radius and all curves a minimum 40-foot radius. The 18-inch-gage track is laid with 40-pound rails and with a grade of 0.3 percent in favor of the load. Switches are interchangeable with those used on 36-inch-gage track for 75-foot radius turnouts.

Haulageways are electrically illuminated. Safety zones, warning lights, and buzzers are provided for the safety of pedestrians. Traffic lights are operated by a car sampler at the ore-pocket station. A switch panel at the station duplicates the lights on the line. Request for clearance is made by the motorman by pull switches from the loading crosscuts.

BLOCK DEVELOPMENT

All-Gravity System

Size of Panel and Block

The purpose of block development is to prepare the ore for extraction. When an all-gravity system - the simplest form of block caving - is used, design is governed by company experience in other mines and experimental operations under local conditions (part 1, p. 14) (fig. 31).

In general, a block is a self-contained production unit of one or more panels. A panel, the smallest mining unit, represents an area served by 1 grizzly drift over 1 haulage crosscut. At the Kelley mine a panel is a rectangular block 80 by 30 feet in horizontal section and is the most suitable for operating efficiency. The 80-foot length was considered to be the minimum safe span over which the rock would cave freely; 16 and 20 feet factors of 80, are close to the most feasible minimum and maximum distances of grizzly spacing. The 30-foot panel width is a well-balanced distance for spacing grizzly drifts and is a multiple of the car length - 10 feet. With crosscut chutes 30 feet apart the 30-foot panels allow train loading from several chutes at the same time and repeated loading by moving the train 10-foot lengths.

The most desirable active area for caving from the standpoint of rapid mining and maintenance is found to be 80 feet wide and 120 or 150 feet long, or a multiple of 4 or 5 panels. The 80- by 120-foot block is considered standard.

The size of the block depends on the size and shape of the ore body, as illustrated by the following examples. In narrow zones a single block may contain 1 to 4

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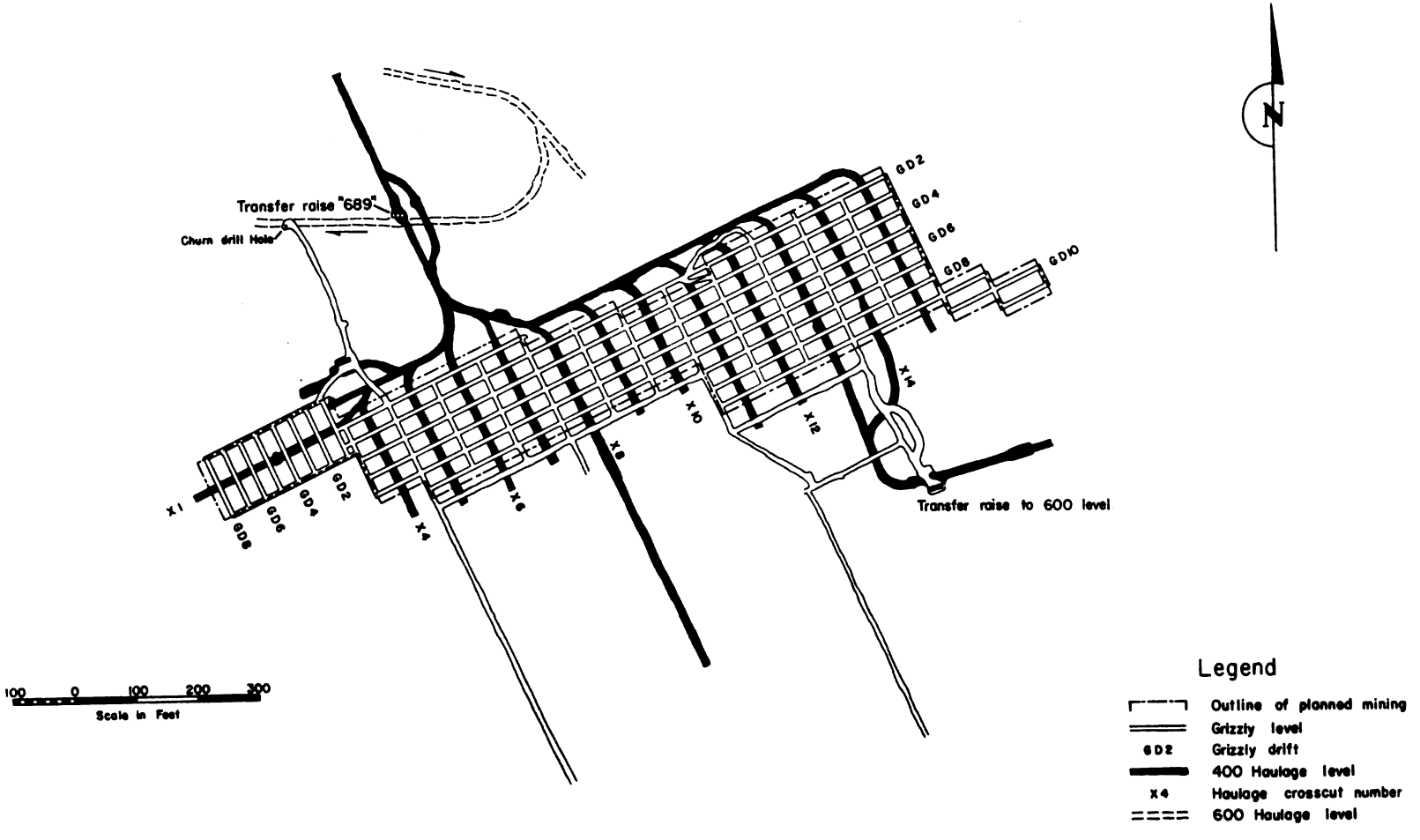

Figure 30. - Zone 9 on 400 level, Kelley mine.

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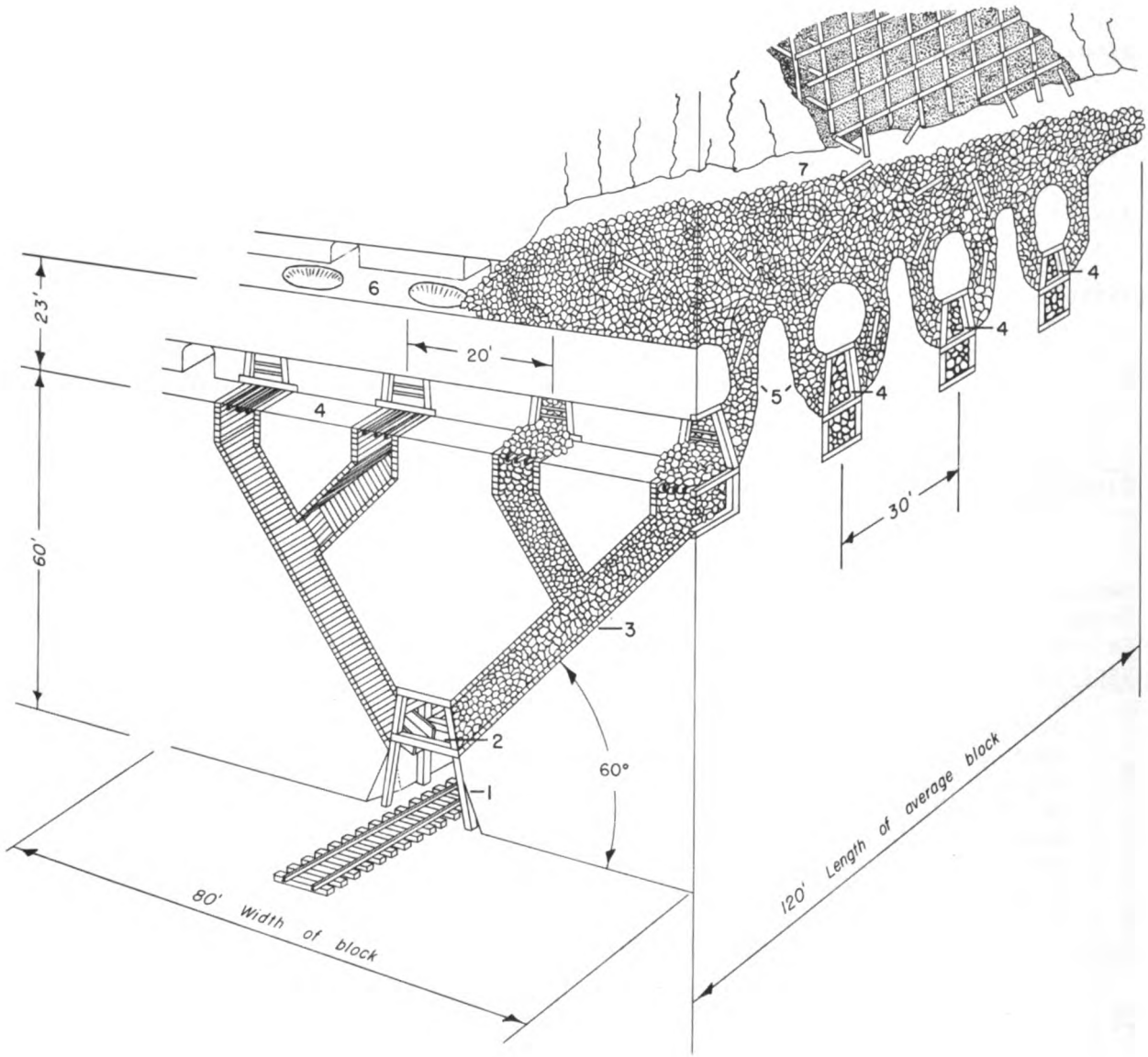

Legend

1 Haulage crosscut
2 Haulage chute
3 Completed raise structure
4 Grizzly drifts
5 Finger raises
6 Undercut level
7 Early stage of caving after pillar extraction

Figure 31. - Block development, all-gravity system.

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panels from the foot to the hanging wall. In wide or irregular ore zones two or more "standard" blocks may be plotted across the width of the ore zone and mined simultaneously; double block areas, however, require more maintenance.

In general, each block has one or more sides exposed. In a block surrounded by four solid sides, little maintenance of the workings is required, and the dilution problem is insignificant. A block with one side adjacent to an exhausted block requires more maintenance and the danger of dilution is increased, particularly if little time is given for waste consolidation in such a block.

The details of standard blocks for the all-gravity system are given in the first column of table 7.

TABLE 7. - Dimensions adopted for standard block by all-gravity and by slusher-gravity systems

	All-gravity block	Slusher-gravity block
Block size:		
Width, 80 feet		
Length, 120 feet		
Area, sq. ft., 9,600		
Haulage crosscuts interval ..... feet	80	
Scraper drift interval ..... do.		60
Grizzly drift interval ..... do.	30	
Vertical distances:		
Haulage level floor-scraper level floor ..		10 ft. 3 in.
Scraper level floor-grizzly level floor ..		24 ft. 10 in.
Haulage level - grizzly level	60 ft.	35 ft. 1 in.
Grizzly level sill-undercut level floor ..	23 ft. to 18 ft.	18 ft.
Grizzly drifts ..... total number	4	4
Distance between fingers ..... feet	12 and 18	12 and 18
Distance between grizzlies ..... do.	20	16
Fingers ..... total number	32	40
Undercut stope area served by 1 finger, ... square feet	300	240

Block Planning and Development

The principal objective of block planning is to lay out the panels and blocks in such manner that they cover the ore zone with a minimum number of mining units. For this purpose a 50-foot-scale map showing the ore boundaries, the average strike, and other pertinent data on each level is prepared. First the ore zone is divided into portions by plotting the length of the panel, 80 feet, along the strike of the zone and by plotting the haulage crosscuts for each portion across the zone (figs. 23, 24, and 30). Cross sections are drawn through each crosscut showing the vertical height of the zone; the 30-foot dimension of the panel is then applied to cross sections to satisfy, as well as possible, the principles of efficient ore drawing and the minimum ore-column requirements for the grade in place (fig. 32). Tonnage and grade estimates are made and recorded for each section in the form of special charts (fig. 33).

Block development is carried out from haulage levels and consists of (a) haulage crosscuts, (b) main raises placed at predetermined intervals and extended to a grizzly level, (c) grizzly drifts, (d) undercut and fringe drifts, and (e) removal

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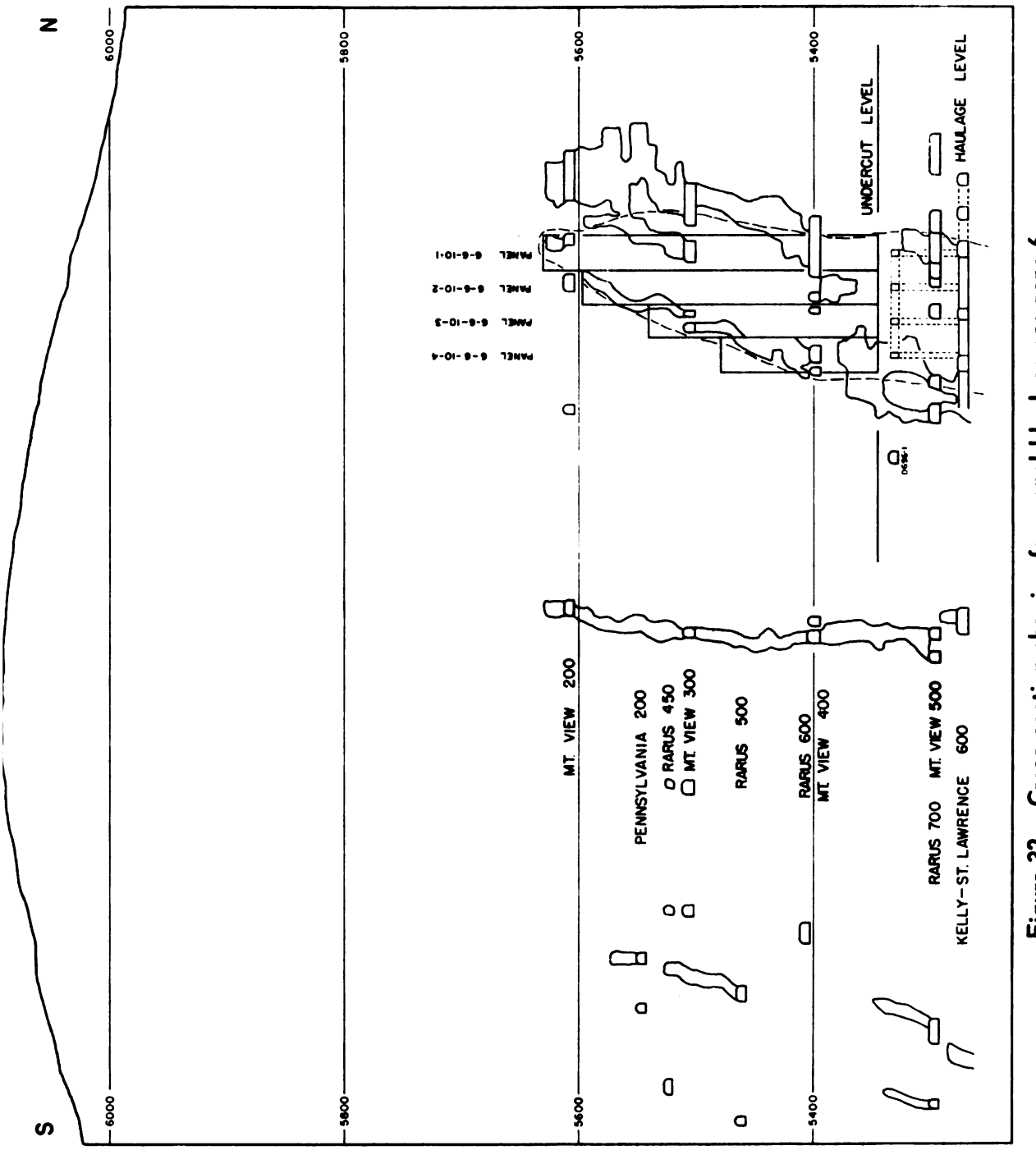

Figure 32. - Cross section showing four-panel block across zone 6.

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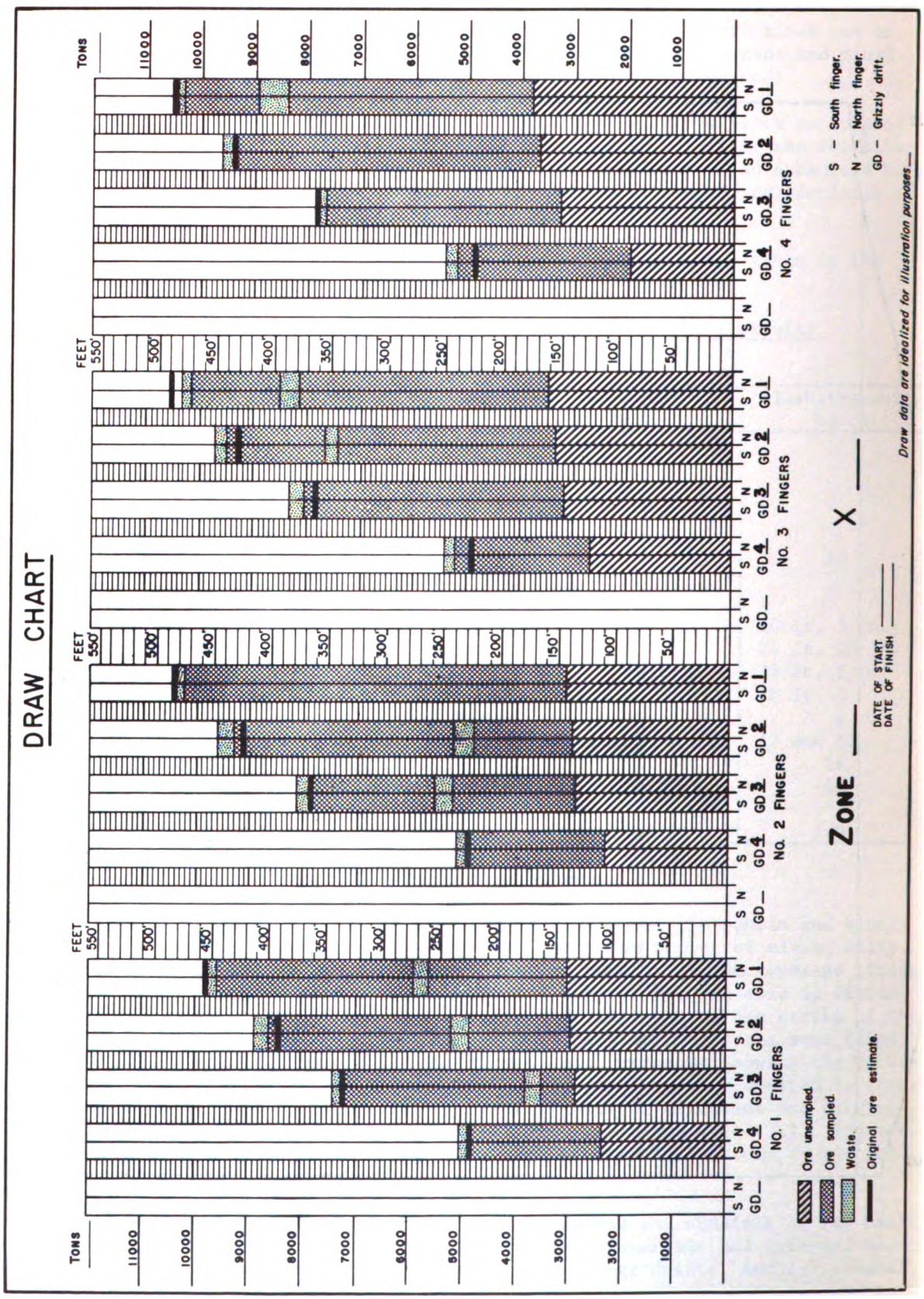

Figure 33. - Draw chart for four-panel block.

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of pillars. This last stage of pillar removal between the undercut drifts removes the support and creates an arch having the dimensions of the block. Ore drawing then causes the arch to fail in tension.

Sequence of Mining Blocks

The sequence of mining blocks and the number of blocks that can be mined at the same time from a given zone depend, in general, on the areal size of the zone and the practicability of maintaining development workings. In a narrow zone, when a single block extends from foot to hanging wall, the original plan was to undercut two adjoining blocks in the center of the zone and to retreat, block by block, from the active caving area in opposite directions along the strike.[20/]

It was found in practice, however, that the initial double block, although surrounded by four solid sides, required 2 or 3 complete changes of grizzly timber support during its productive life. The active caving area (160 by 120 - 150 feet) was too great. Better operational conditions were obtained and less timber maintenance was required when a new block was separated from the double initial block by an 80-foot pillar. A block adjacent to unconsolidated waste of an exhausted block required less timber maintenance than the initial double block but more maintenance than the 80-foot block with 4 solid sides.

As a result of experience, all-gravity mining in each zone is now planned in such manner that caving areas do not exceed 80 feet along the strike and 120 to 150 feet across the ore zone (fig. 23); as far as possible, each new block has no more than 1 side exposed to an exhausted block, and a block exposed on more than 1 side is placed into production after a period of waste consolidation of at least 3 months in the adjoining depleted block.

Zone Haulage Crosscuts

Zone crosscuts are designed to permit locomotive haulage and the use of long trains without obstructing the main laterals. The crosscuts for zone 9 on the 600 level and crosscuts for zone 1 on 1,300 and 2,000 levels are parts of clockwise-loop haulage systems. For zones 6 and 8 on the 600 level and for auxiliary levels the zone crosscuts are dead-ended a convenient distance beyond the chutes.

Chutes

Haulage chutes with 36-inch gates are spaced 30 feet apart to allow multiple car loading. This width, three times the maximum size of ore passing the grizzly, was adopted as standard to assure free running and proper control of ore flow when loading into cars.

Originally, Inspiration-type arc gates and Mountain Con-type double gates, both operated by hand, were used (figs. 34, 35, and 36). Experience and special tests showed, however, that mechanically operated gates are more efficient than hand-operated gates; the loaders have a free hand to use a bar. The air-operated undercut and overcut arc gates are shown in figures 37 and 38; air-operated undercut guillotine-type gates are illustrated in figures 34 and 39. Because of high initial cost, the more efficient guillotine-type gate is used for chutes serving a high ore column.

20/ Hannifan, M. K., The Greater Butte Project: Min. Cong. Jour., June 1955, pp. 46-49.

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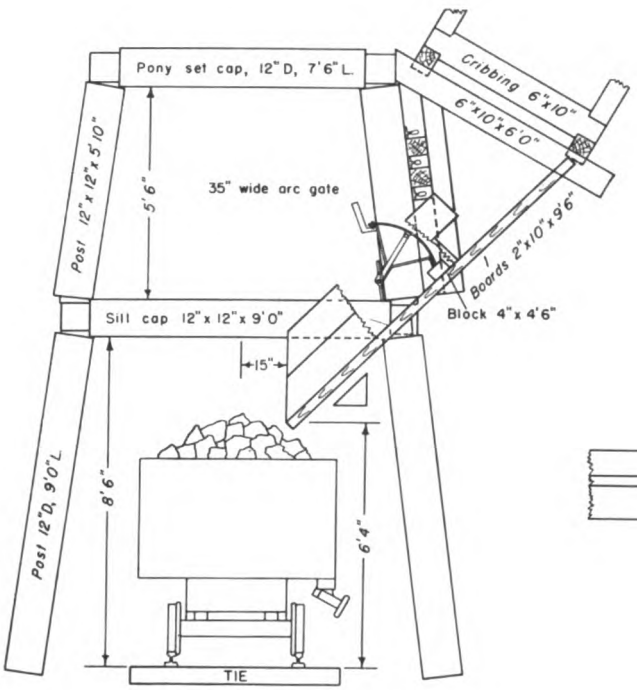
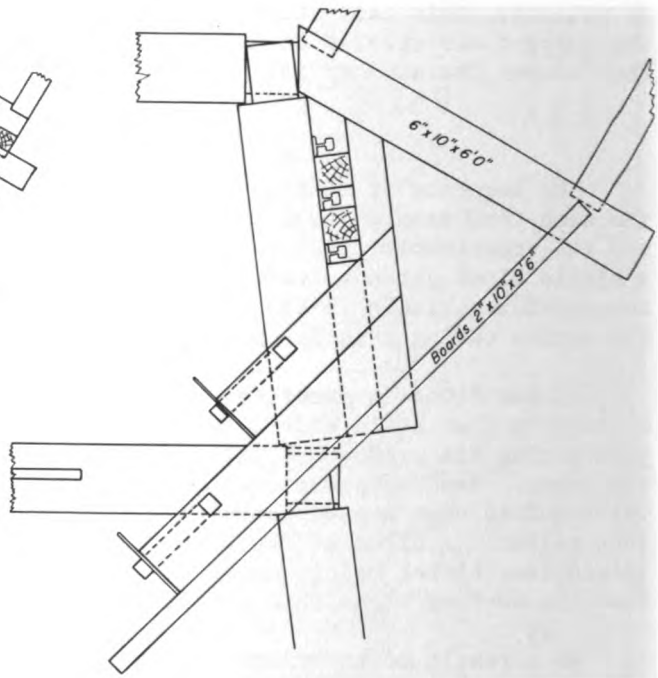
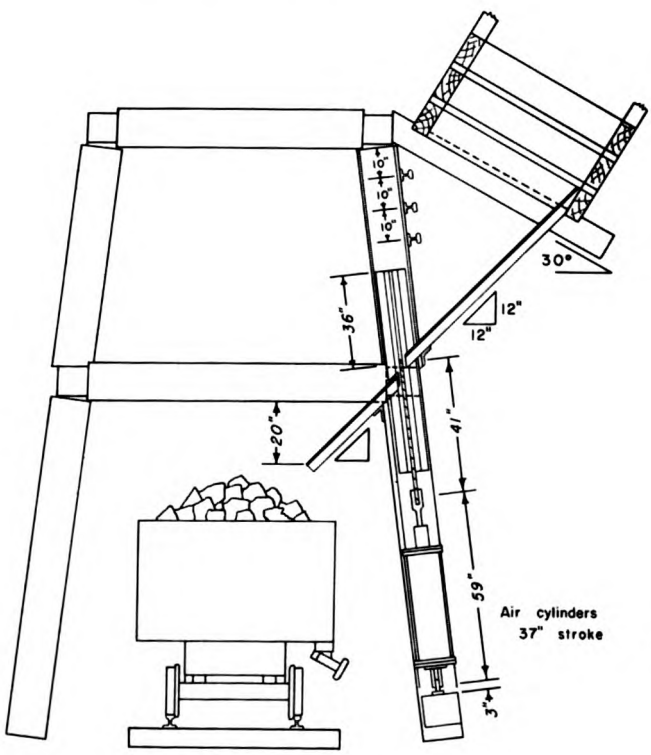


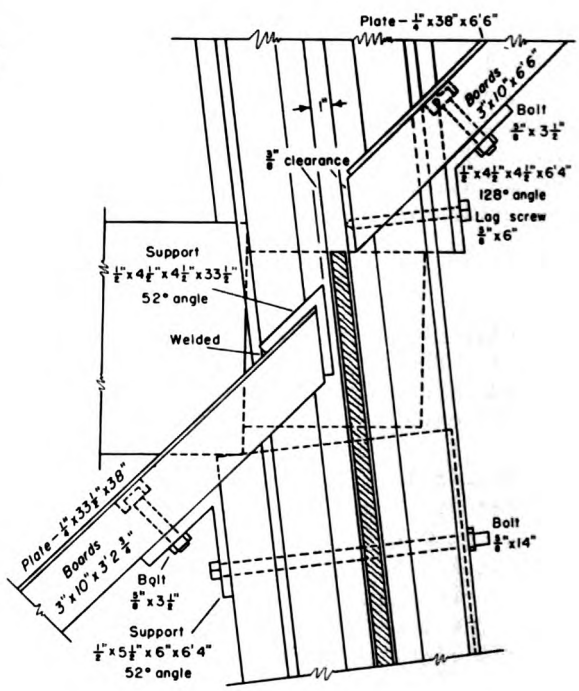
INSPIRATION MINE TYPE



CHUTE DETAIL, DOUBLE MT. CON TYPE



UNDERCUT GUILLOTINE TYPE



CHUTE DETAIL, UNDERCUT GUILLOTINE TYPE

Figure 34. - Chutes and gates, all-gravity system.

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Figure 35. - Inspiration-type arc gate.

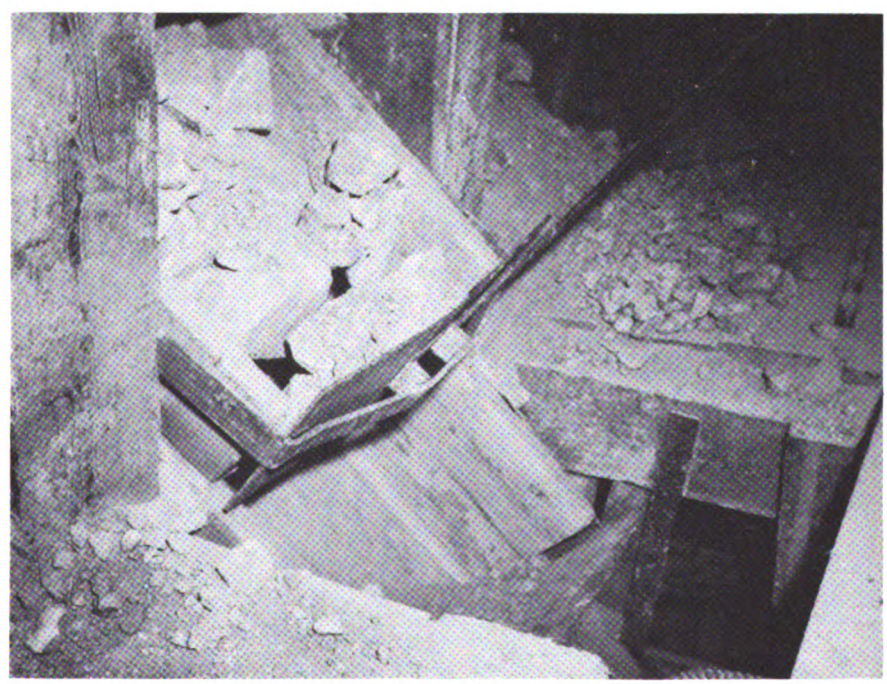

Figure 36. - Mt. Con-type gate.

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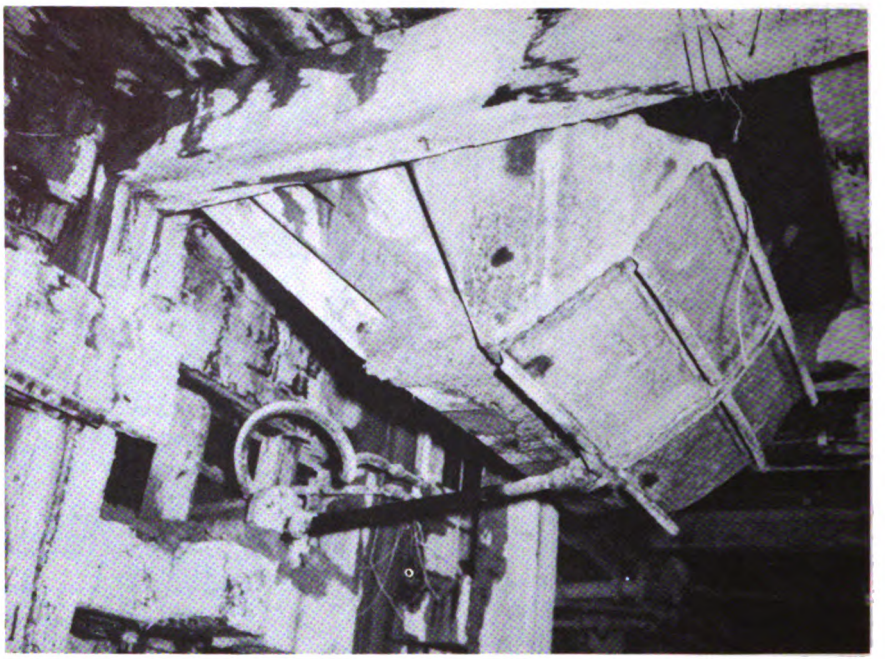

Figure 37. - Air-operated undercut arc gate.

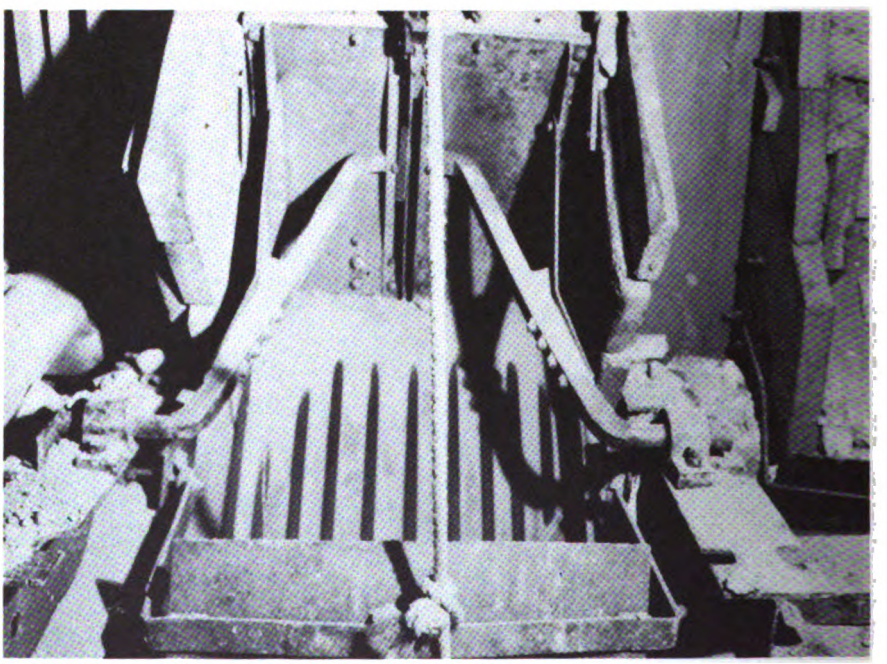

Figure 38. - Air-operated overcut arc gate.

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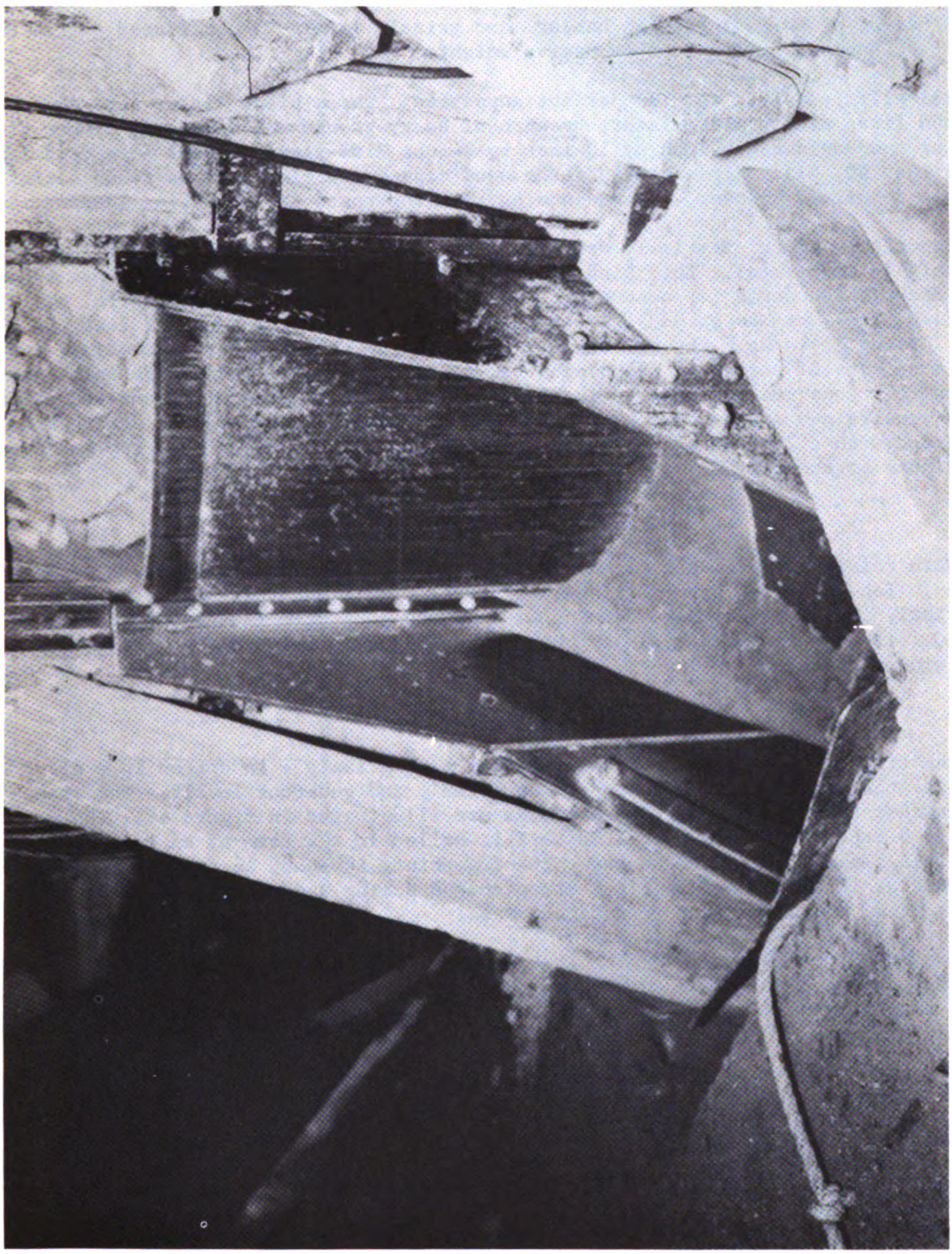

Figure 39. - Air-operated guillotine-type gate.

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Grizzly Drifts

A height of 60 feet between haulage and grizzly levels is considered sufficient to protect haulage drifts from excessive weight.

All grizzly drifts are 100 percent supported. Three-piece timber sets were reinforced first with timber; later, because of heavy ground and damage to timber from blasting and boulders, 8-inch by 10-inch by 8-foot wide-flanged beams were used for caps (figs. 40, 41). Replacement of the wood caps by steel beams was found satisfactory; it required, however, maintenance by changing posts and relieving lagging.

Concrete support of grizzly drifts, first introduced in the spring of 1953, was found highly successful. This kind of support now is accepted widely for new blocks. The prefabricated forms for concreting consist of 4 pieces 5 feet long (fig. 42). Installed, they form an arched drift 4 feet wide at the bottom by 6-1/2 feet high (figs. 43 and 44). When the grizzly drifts are being prepared for concreting, two-piece arch sets of used rail are placed for temporary support where rock bolts are inadequate. Concrete averages 18 inches in thickness. In general, concreted drifts do not require any maintenance. If major cracks develop, however, the drifts are reinforced with a 2-piece arch set made of 60-pound rail bolted at the top and spread with timber at the bottom.

Main Raises

Main raises are driven from the haulage level to the grizzly level on a 60° incline. Together with branch raises, the system forms openings 20 feet apart on the grizzly levels (fig. 45). Continuous 6- by 10-inch crib lining 4 by 4 feet in the clear is held in place by 6- by 6-inch wide-flange steel beams at the branch raise window opening.

Finger Raises

The finger raise system - two raises for each grizzly - is driven in a plane normal to the grizzly line and connecting grizzly and undercut levels. The stages of advancement for an average finger raise are illustrated in figure 46. The raises are 4 to 5 feet in diameter; the top half is belled in the form of an inverted cone with sides sloping 60°; and the base, the drawpoint, is about 16 feet in diameter at the undercut level. The drawpoints are prepared during undercutting. Each drawpoint serves 300 square feet of undercut area.

When the grizzly drifts are concreted, the finger-raise openings are likewise concreted, using three-piece, prefabricated wood forms shaped to fit the arch of the drift; the 1-foot linings are reinforced by a steel beam in the brow of the opening (fig. 47).

Undercutting

Requirements

The ore is undercut to create roof weakness that causes caving. Efficient undercutting requires minimum ground support and minimum mucking. As a result of extensive research by the company many improvements have been made in the original procedure. Rapid, complete undercutting eliminates the need for ground support, brings blocks into production sooner, increases efficiency of the mining crew, and decreases both loss of ore and dilution.

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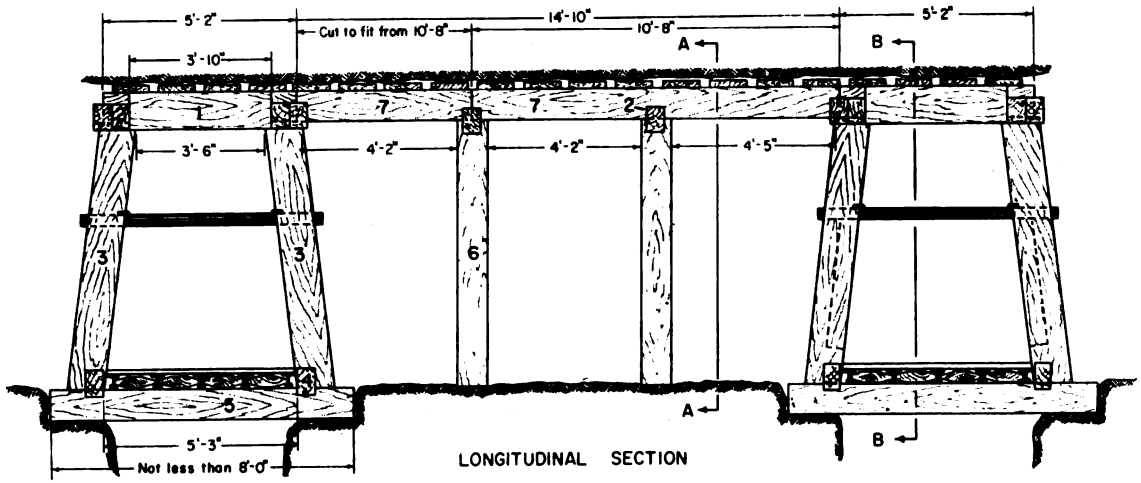
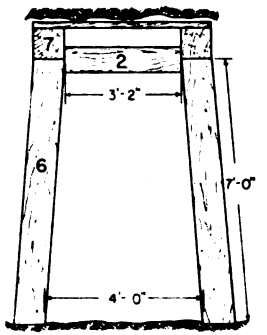


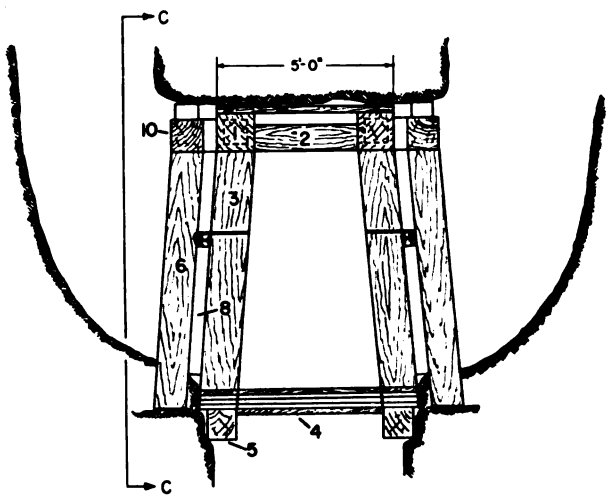


SCALE IN FEET

0 2 4 6 8 10

TIMBER DIMENSIONS

NO.	NAME	DIMENSION
1	Grizzly set cap	12" x 12" x 5'2"
2	Collar brace	6" x 8" x 3'2"
3	Grizzly post	12" x 12" x 7'0"
4	Grizzly spreader	6" x 8" x 4'0"
5	Grizzly sills	10" x 10" x —
6	Drift post	10" x 10" x 7'0"
7	Drift stringer	10" x 10" x —
8	Rail support	4" x 6" x —
9	Control set post	10" x 10" x 7'0"
10	Control set cap	10" x 10" x 11'0"



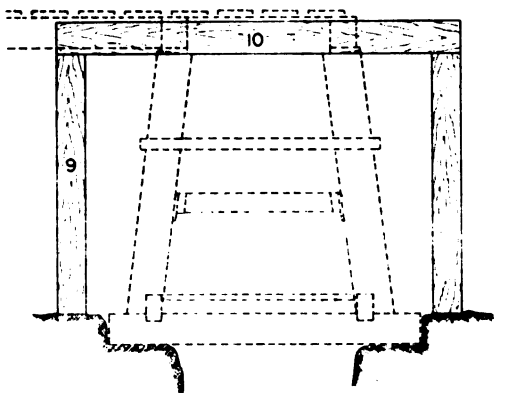


Figure 40. - Grizzly drift timbering, all-gravity system.

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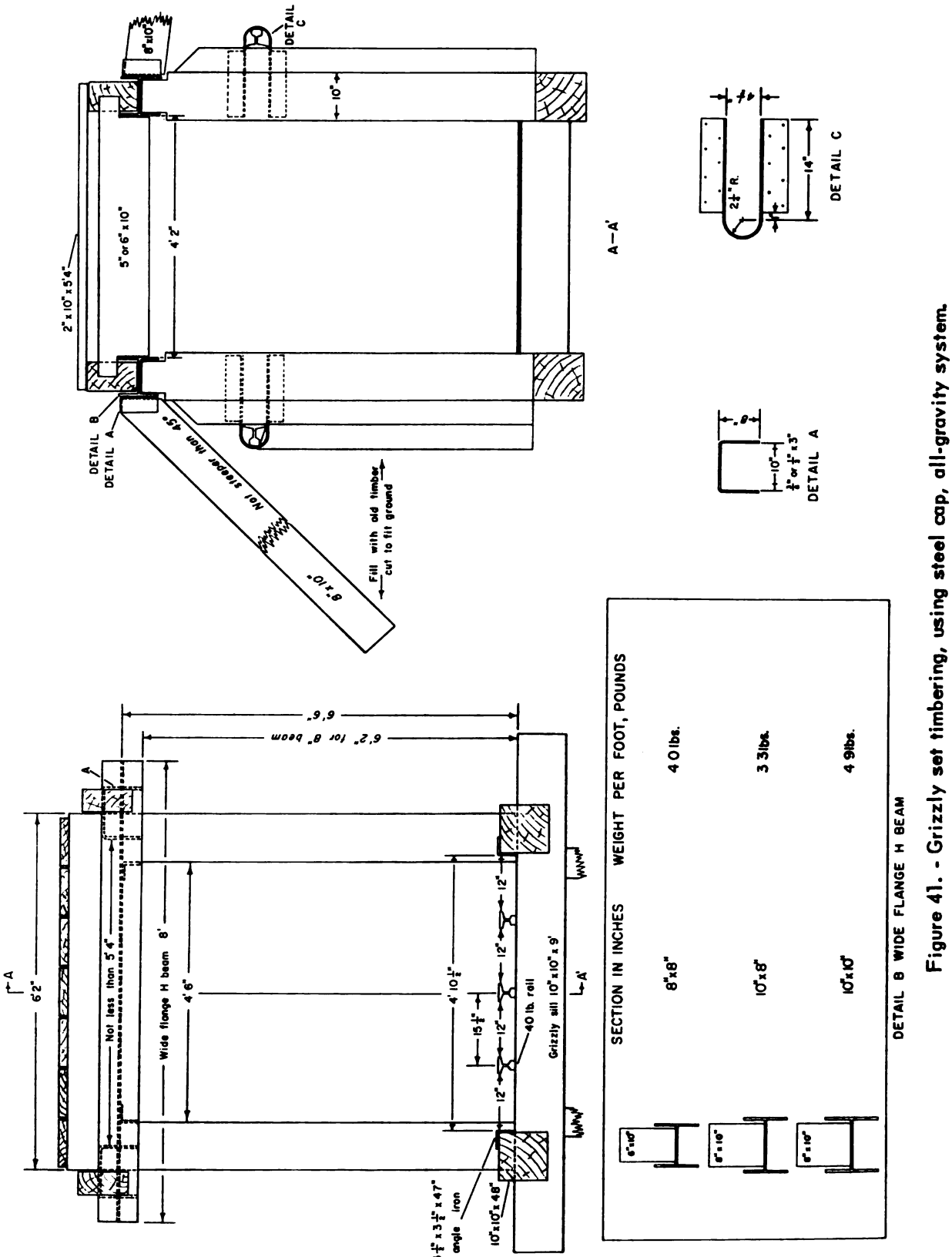

Figure 41. - Grizzly set timbering, using steel cap, all-gravity system.

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Figure 42. - Wood concrete forms installed for grizzly drift and finger raise.

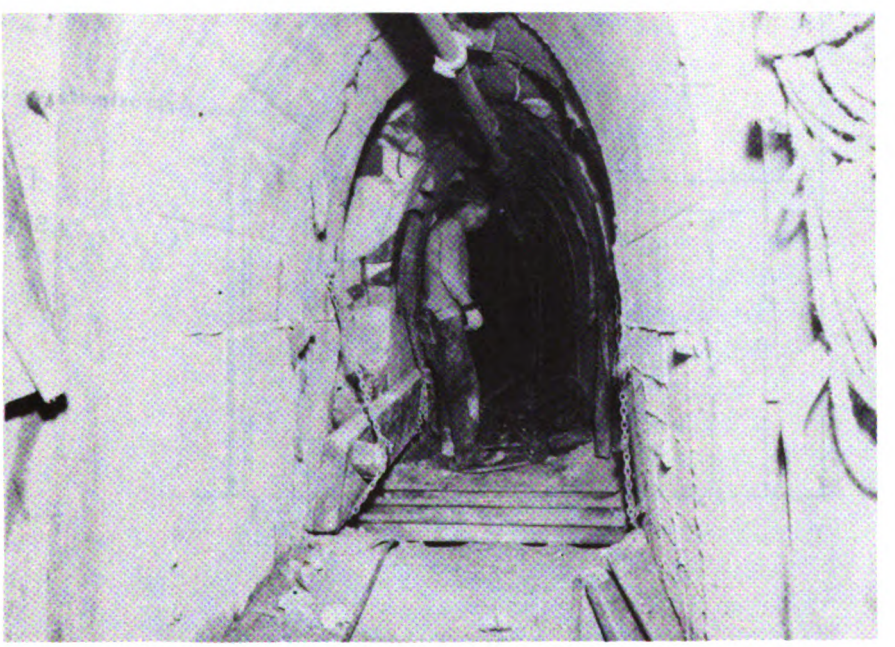

Figure 43. - Arched concrete grizzly drift.

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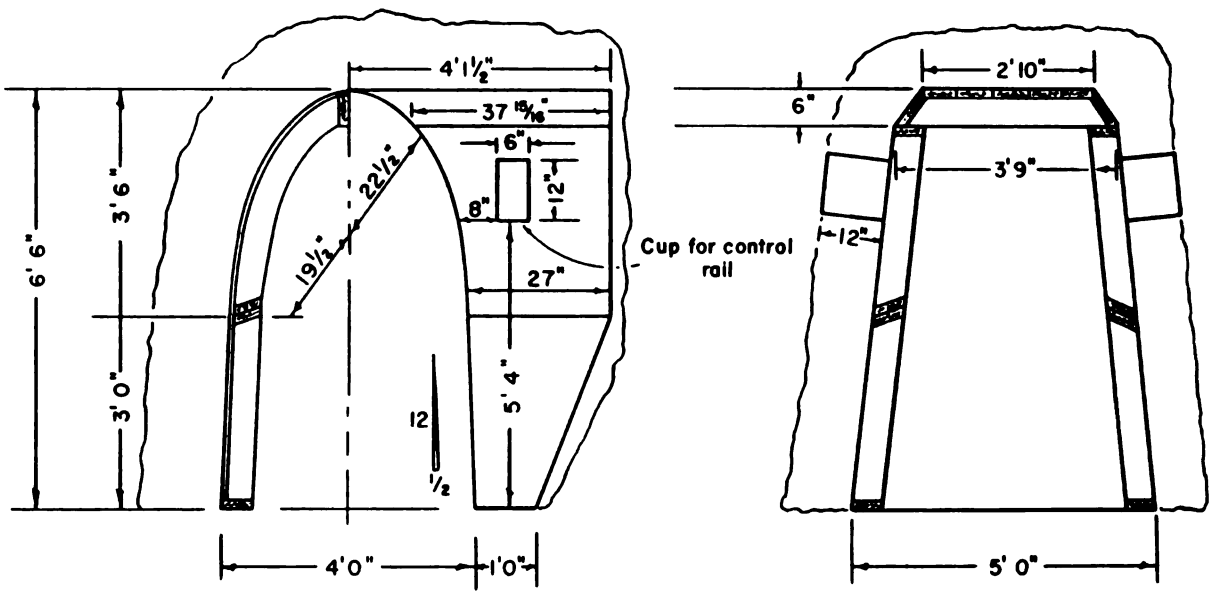
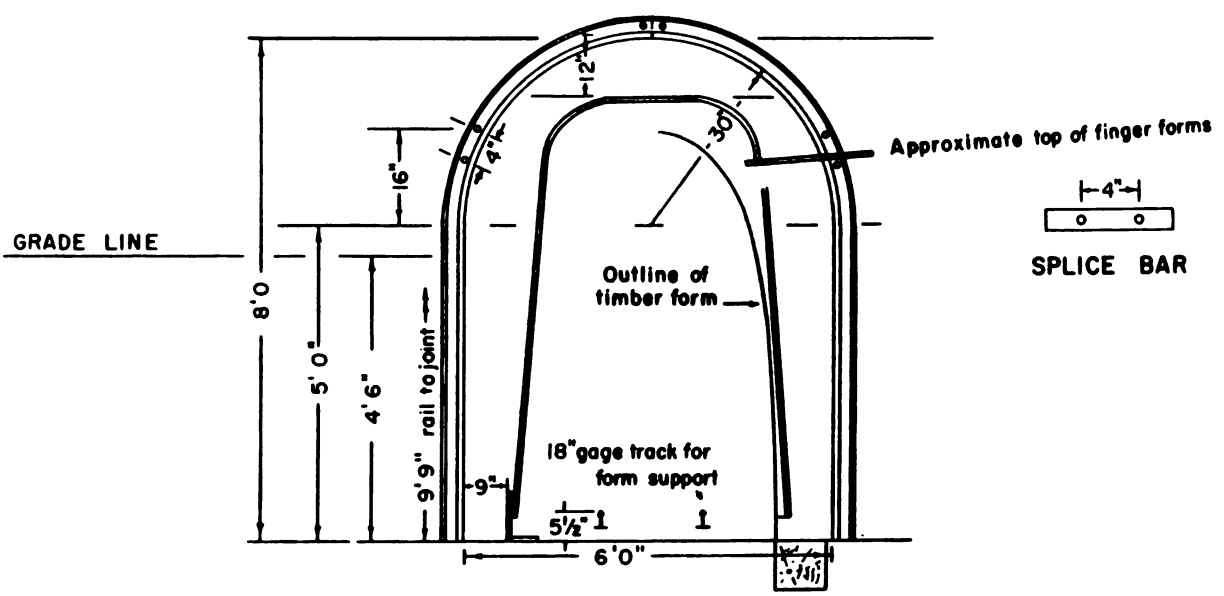


WOOD CONSTRUCTION FORMS



STEEL CONSTRUCTION FORMS

Figure 44. - Grizzly drift, concrete support.

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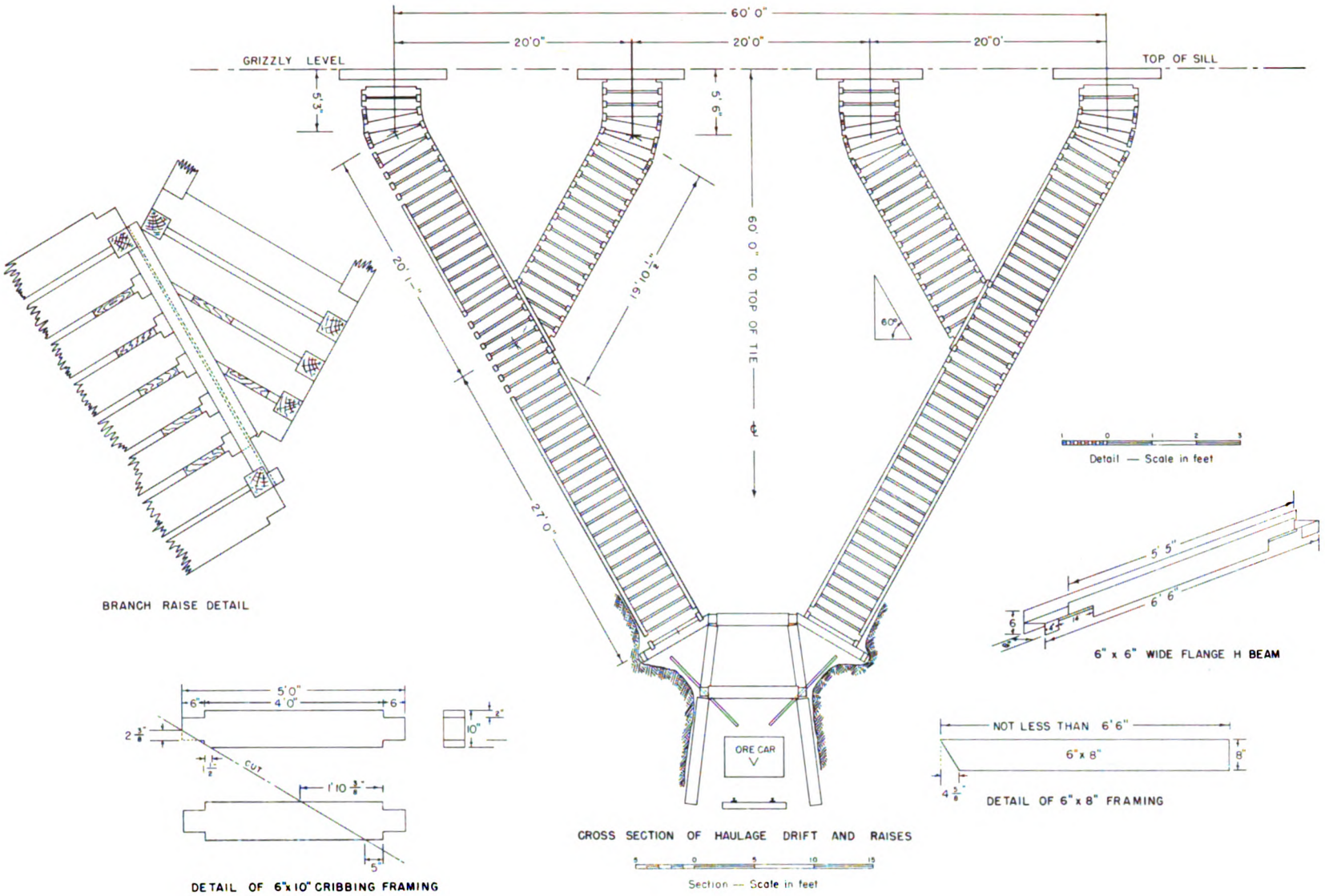

Figure 45. - Main raises, all-gravity system.

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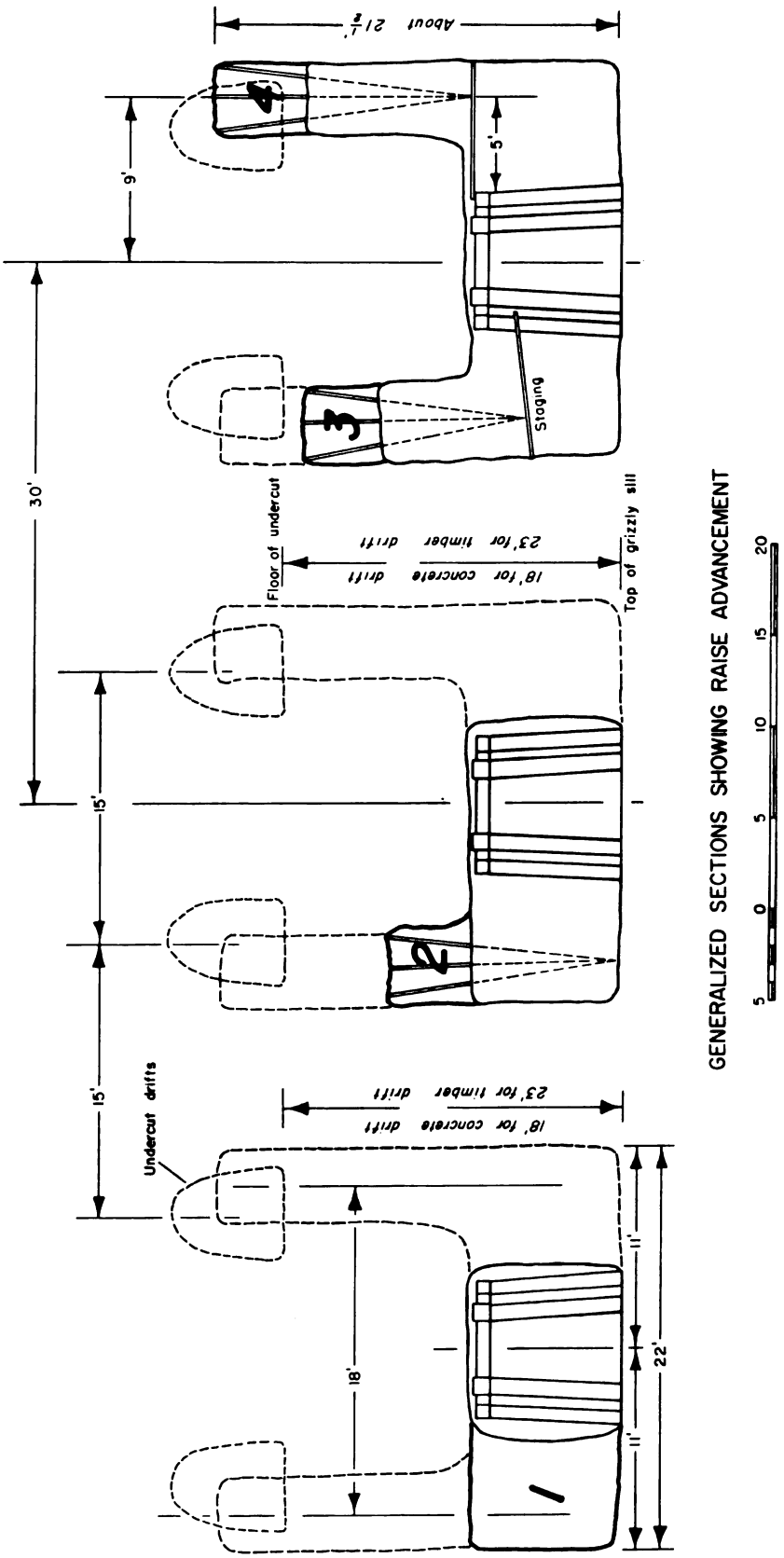

Figure 46. - Finger raises, all-gravity system.

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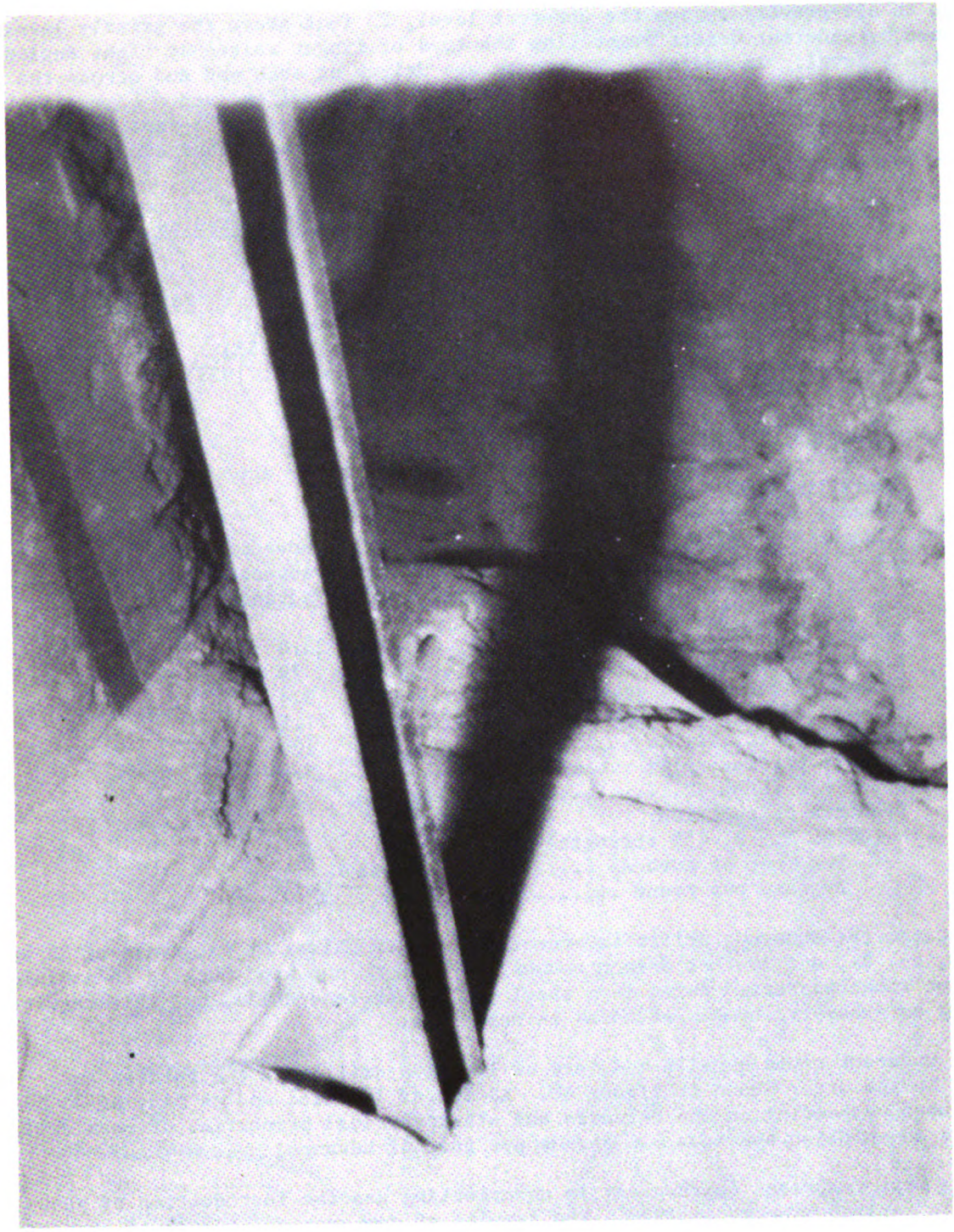

Figure 47. - Concreted finger raise opening with steel beam at brow and control rail in cup.

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Undercut Drifts

In an all-gravity system the undercut level, 23 feet above the grizzly level, is opened by undercut drifts connecting the tops of finger raises at right angles to the grizzly level. In addition, crosscuts are driven at each end and across the center of the block (fig. 48). In all, 4 drifts, 2 fringe drifts, and a communication crosscut are usually driven for each block. The drifts are 7 by 5 feet in cross section. The average advance is 2.7 feet per man-shift; the record advance is 5.4 feet per man-shift (table 8).

TABLE 8. - Undercutting; labor output and time studies

1. Undercut drifts - labor output

	Feet per man-shift
Advance, record (see time study below for details)	5.4
Cumulative average for zone 6, (July 1, 1953)	2.7
Cumulative average for all zones, (July 1, 1953)	2.7
Stopers only	2.3

2. Undercut drifts - time studies and other pertinent data using push-feed drill, two shifts per day

Operating days - 24
Advance in feet - 563

Rounds:	5-foot rounds	38, total footage advance	190
	8-foot rounds	24, total footage advance	192
	10.7-foot rounds	17, total footage advance	181
	Total	79	563

Advance per operating day	feet	23.4
Number of blasts	total	79
Number of blasts per operating day		3.4
Average advance per blast	feet	7.1
Man-shifts		104
Advance per man-shift		5.4
Holes per round	average	17
Dynamite consumption:		
(Gelax No. 2, 45% strength)		
Per foot of advance	pound	4.6
Average per round	do.	33.0

Because the undercut drifts intersect the finger raises at intervals of 12 and 18 feet, 5-, 8-, and 10-foot 8-inch rounds are used to cut hand mucking to a minimum. A 16-foot round was tried using long steel and a push feed drill, but this required drilling an excessive number of holes in hard rock.

An undercut round usually consists of 16 holes drilled in the pattern and blasted in the order shown in figure 48. As few as 10 holes and as many as 25 holes may be used, depending on the hardness and other physical properties of rock drilled. Dynamite consumption averages 4.6 pounds per foot of advance.

The most important improvement in undercutting was the introduction of push-feed drills to replace stopers in 1952. The results of comparative studies of stopers and push-feed drills are given in table 9.

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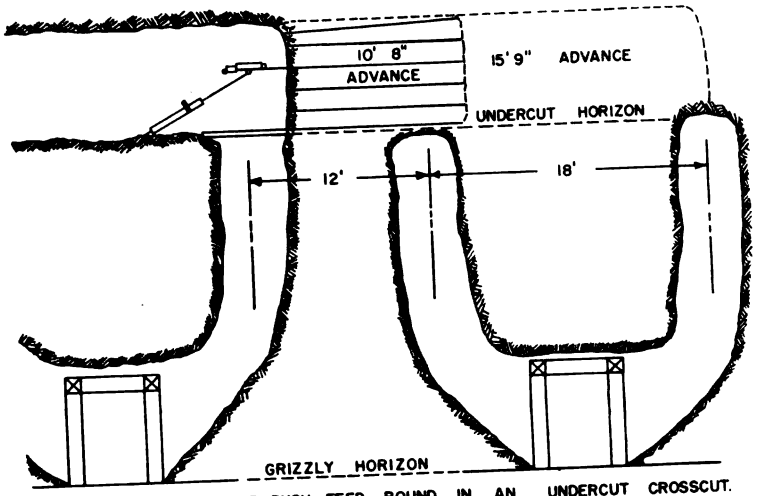
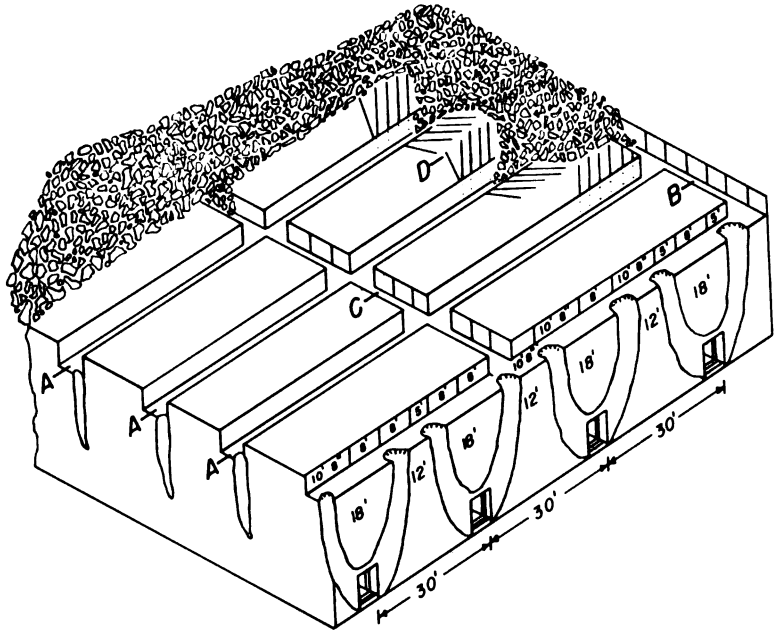
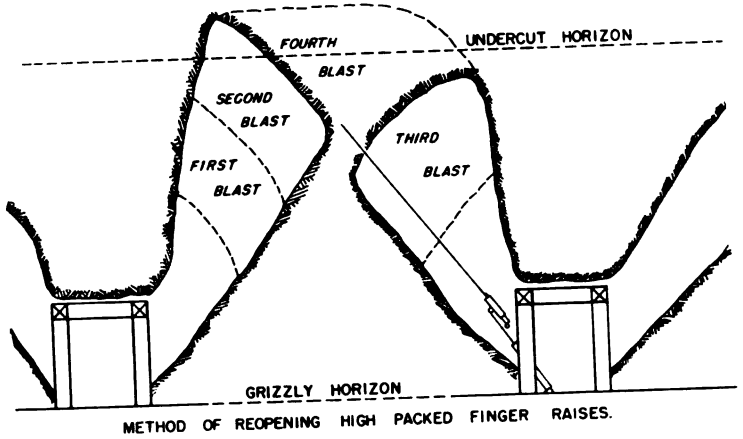


LONGITUDINAL SECTION OF PUSH FEED ROUND IN AN UNDERCUT CROSSCUT.



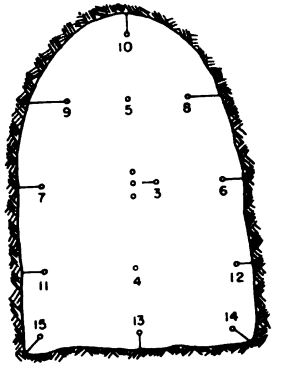
BLOCK DIAGRAM



METHOD OF REOPENING HIGH PACKED FINGER RAISES.

BLOCK DIAGRAM LEGEND

A. Undercut drifts, 10' 8", 8' and 5' push feed rounds.
B. Fringe drift.
C. Communication crosscut.
D. Herringbone pattern of holes drilled for pillar extraction.



16 HOLE ROUND, 5'x7' DRIFT.

Figure 48. - Undercutting, all-gravity system.

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TABLE 9. - Undercutting - stopers versus push-feed drills

	Stoper drill	Push-feed drill 1/
Round advanced ..... feet	6-8	11
Holes ..... depth in feet	8	11
Steel changes	5	4
Steel changes in inches	18	31
Drill speed ..... inches per minute	37.9	26.3
Overall rate per man ..... minute per foot	0.90	0.62
Time reduction ..... percent		30
Rate per man-shift ..... feet	2.3	5.4
Man efficiency increase ..... percent		235
Standard drill-rod lengths, inches		
	Stoper drill	Push-feed steel
Starter, for 2-foot 9-inch hole	39	39
Second, for 4-foot 4-inch hole	59	
Third, for 6-foot hole and rock bolting	79	79
Fourth, for 7-foot 8-inch hole	99	
Fifth, for 9-foot plus hole	119	119
Sixth, for 11-foot plus hole	139	139

1/ With 36-inch leg: 48-inch leg too long for small undercut workings; 24-inch leg require more steel changes.

Although push-feed-drill speed (26.3 inches per minute) was nearly 45 percent slower than stoper-drill speed (37.9 inches per minute), the overall efficiency of labor was more than twice as great because of lower labor fatigue and because of the convenience in use of the machine.

Pillar Extraction

The pillars, 15 feet wide, are removed in retreating operations by drilling rounds in herringbone pattern from undercut drifts and blasting the holes with dynamite. The removal proceeds as fast as possible. Inspections are made after each round to be sure that no pillar left on the level will throw excess weight on the workings below.

Labor output and time studies of pillar extraction in table 10 show that a crew of 2 men averages two 400 drill-foot rounds per shift and produces an average of 70 square feet of pillar extraction per man shift.

Slusher-Gravity System

General

The advantages of concrete support for block development made tests with a combination slusher-gravity mining system a success. Consequently, this system is being used on an ever-increasing scale above the 600 level and exclusively in the 1,300 level.

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TABLE 10. - Undercutting - pillar extraction, labor output and time studies

1. Labor output

	Square feet per man-shift
Performance, record (see below for details)	142.0
Cumulative average for zone 6 (July 1, 1953)	98.5
Cumulative average for all zones (July 1, 1953)	70.0
Stopers only	58.0

2. Estimate of drilling per shift (block 6, zone 6)

a. Estimate of drilling for one shift

Men - 2 Rounds - 2 Undercut area - average, 230 sq. ft.

	Undercut drifts	Fingers (belling)	Total
Holes per miner	42	14	
Hole depth, feet	8	5	
Total feet per man	336	70	406
Drilling speed man-minute per foot, minimum (see studies below)			1.00
Total time required per man for drilling			6 hr. 46 min.

b. Time studies: Time required for drilling one round (400 feet)

Drill speed, inches/min.	Drilling rate, man-min./foot	Total time required for 1 round
15.1	0.95	6 hr. 20 min.
25.0	.86	5 hr. 44 min.
25.5	.82	5 hr. 28 min.
30.2	.53	3 hr. 32 min.
31.2	.53	3 hr. 32 min.

c. Summary for block

Operating days - 12
Total square feet - 9,488
Square feet per man-shift - 142
Square feet per blast - 791
Total holes - 2,118
Total linear feet - 16,944
Average undercut per hole - 4.5 ft.
Cartridges per hole - 8.5
Powder consumption - 0.525 pound per square foot

In the slusher-gravity system the caved ore moves by gravity to the slusher drift, driven below the grizzly level, and the ore is transported and loaded into cars by scrapers. Early block-design details are given in table 7 (p. 46). The undercut-stope area served by 1 finger is 240 square feet in the slusher-gravity block compared with 300 square feet in the all-gravity system. The slusher drifts, driven above the cap and normal to the haulage drift, and the grizzly level drifts are concreted to support the overlying ore and to resist the wear and tear of rock pounding and blasting.

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Development workings required for slusher-gravity and all-gravity blocks of similar size are compared in table 11 and illustrated in figure 49. It is estimated that the development cost of a slusher-gravity block is 16 percent higher than that of an all-gravity block.

TABLE 11. - Required development for standard block by all-gravity and by slusher-gravity systems

	All-gravity system		Slusher-gravity system
Class of work	Per panel (30- by 80-foot)	Per block (80- by 120-foot)	Per block (80- by 120-foot)
Haulage crosscut..... feet	45.3	181	181
Scraper drift..... do.			230 (concrete)
Main raises..... do.	160	640	400
Grizzly drifts..... do.	110	440 (concrete)	440 (concrete)
Finger raises..... do.	160	640	800
Undercut..... do.	180	720	720
Cost per square foot of required development. ..... percent		100	116

Basic development data

	All-gravity system	Slusher-gravity system
Scraper drift interval.....feet		60
Grizzly drift interval..... do.	30	30
Area of finger.....square feet	300	240

Advantages

The slusher-gravity system has been proved to be more efficient because it allows a higher rate of production. Ore is fed from the grizzly level through shorter raises that are less apt to hang up and are more easily freed when plugged. Chutes and chute gates, with their costly repairs, are eliminated. Car loading by scrapers is uninterrupted by maintenance on the grizzly level or in raises, proceeds with less spillage, and, in general, is controlled more easily. Closer spacing of drawpoints allows more rapid and uniform drawing, better handling of wet, sticky ore, and, probably, better overall ore extraction. Larger ore can be handled by this system, permitting wider grizzly rail spacing.

Concrete support of workings permits development to be carried on without danger of timber decay and failure many months before drawing. Tramming, handling, placing bulky timber, and expensive timber maintenance are eliminated. Concrete lining of slusher drifts provides better working conditions for scraper loading; concreted fingers allow better flow of ore.

The concreted workings in the slusher-gravity system provide better ventilation, less fire and dust hazards, and less damage from blasting and boulder blows in comparison with timber-supported workings in the all-gravity system. Labor efficiency increases considerably. The output per man-shift has increased from 45 tons to 97.

The slusher-gravity system has been used successfully where the all-gravity system was unsatisfactory, such as in a double block area (160 by 120 feet) or in a block lying between the foot and hanging walls and adjacent to depleted blocks on two sides.

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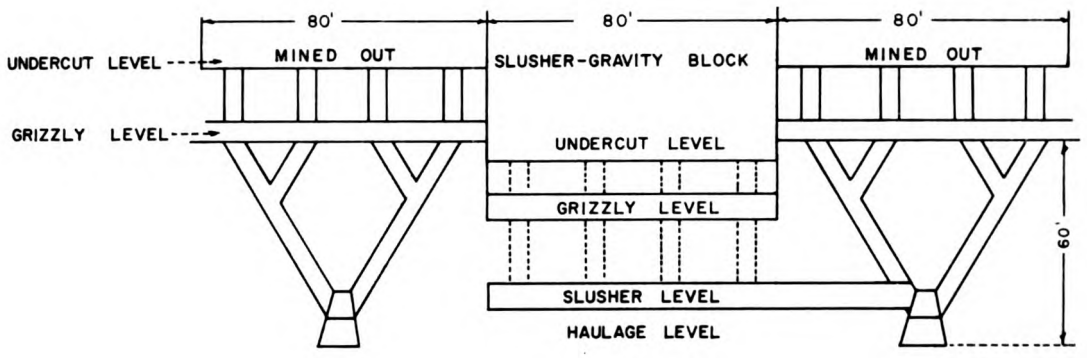
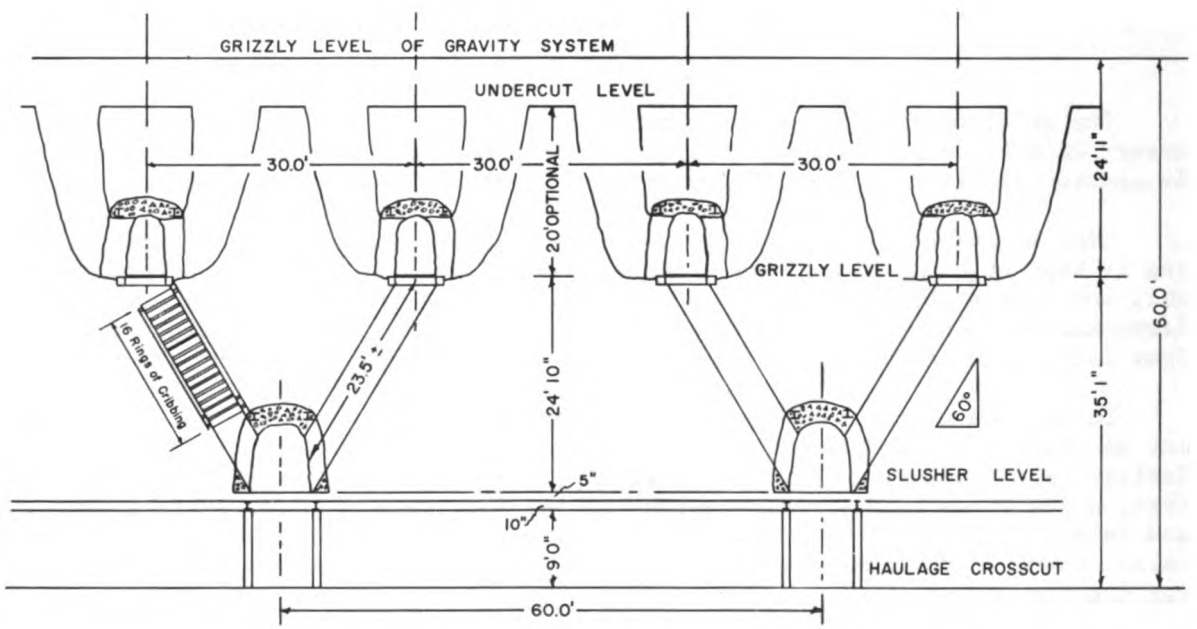


A. LAYOUT BETWEEN ALL-GRAVITY BLOCKS.



B. SLUSHER DRIFT PARELLEL TO GRIZZLY DRIFTS

Figure 49. - Block development, slusher-gravity system, sheet 1.

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Slusher Drifts

The standard, concreted, slusher drift is 9 feet 4 inches high and 7 feet 3 inches wide (figs. 50 and 51). The present practice is to use -75 hp., 2-drum electric slushers with a rope pull of 9,900 pounds and a speed of about 250 feet per minute (fig. 52). The bucket is 60 inches wide (fig. 53). Total shift production is 500 to 700 tons per slusher drift.

Variations in Design

In the early design of a slusher-gravity block, slusher drifts were parallel to the grizzly drifts, raises were driven upward on a 60° angle, and the finger raises were nearly vertical (fig. 49,B). The design was later modified to eliminate packing in main raises; the interval between grizzly and slusher levels was shortened and a vertical section under grizzlies was added to the main raises (fig. 54,A). In the third type of block design slusher drifts are transverse to the grizzly drifts, grizzly intervals are 20 instead of 30 feet, and special crosscuts are added for ventilation (fig. 54,B). In a more recent design, three branch raises provided better draw control; the slusher drifts are 50 feet apart and transverse to the grizzly drifts. This design also includes ventilation crosscuts (figs. 55,A, and 56).

Tests are being made with a new design where slusher drifts are 30 feet apart and no grizzly drifts are provided (fig. 55,B).

ORE DRAWING

Rate and Distribution

Ore is drawn from fingers at whatever uniform rate will achieve maximum ore recovery with minimum dilution. This rate fluctuates during the life of the block and is controlled mainly by the caving behavior and physical character of the ore.

Time is allowed for caving to take place after undercutting. Thereafter, drawing is kept at a curtailed rate to get good caving action so that the broken ore will not support the caving ground and stop the sloughing or, on the other hand, leave excess open space in the stope, objectionable because of possible air blasts from falling large masses.

In subsequent stages caving proceeds at a fast rate, and the speed of drawing may be increased according to production needs, limited only by available technical facilities and labor. If drawn too slowly, the clayey ore tends to pack and consolidate, which transfers excessive weight to the workings, requires extra maintenance, and delays the underground flow of ore. In fast yet nonuniform drawing the cave material tends to roll and slide, channels waste to the drawpoints, and gives misleading drawing data.

The general procedure is to draw uniformly from all fingers and to decrease the rate of drawing from raises in which dilution appears. If, in spite of slower drawing, dilution increases, all diluted draw raises are stopped temporarily to allow consolidation of waste channels in the ore. These raises are again reopened then drawn slowly and sampled. Such procedure may be repeated several times during the life of the block.

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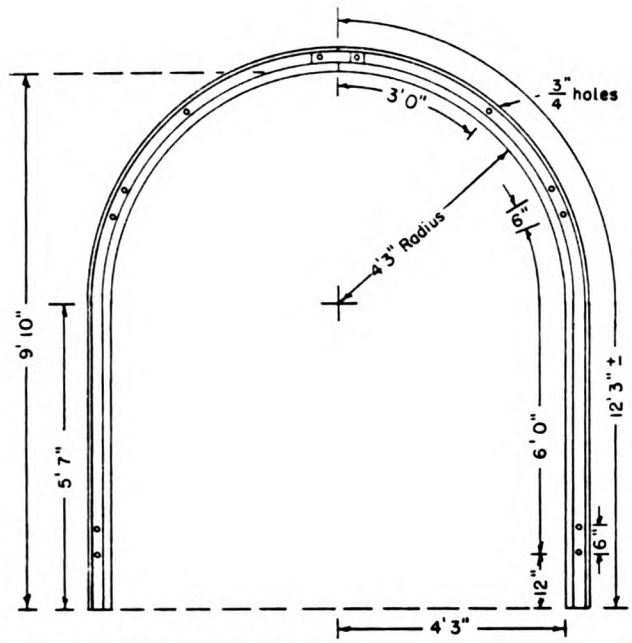
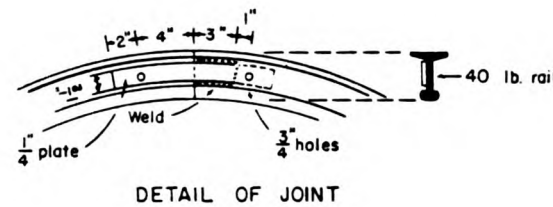
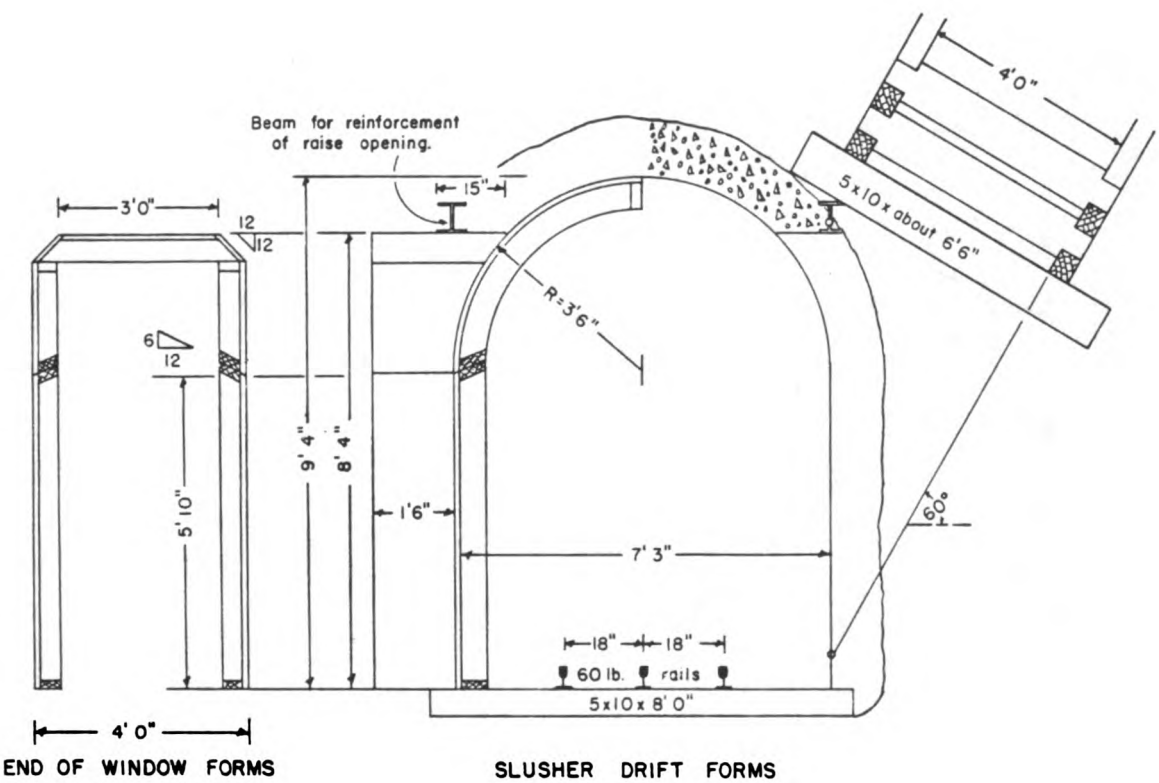


SLUSHER DRIFT REINFORCED BY TWO-PIECE RAIL SET



DETAIL OF JOINT



END OF WINDOW FORMS

SLUSHER DRIFT FORMS

Figure 50. - Seven-foot slusher drift, wood construction forms for concrete support.

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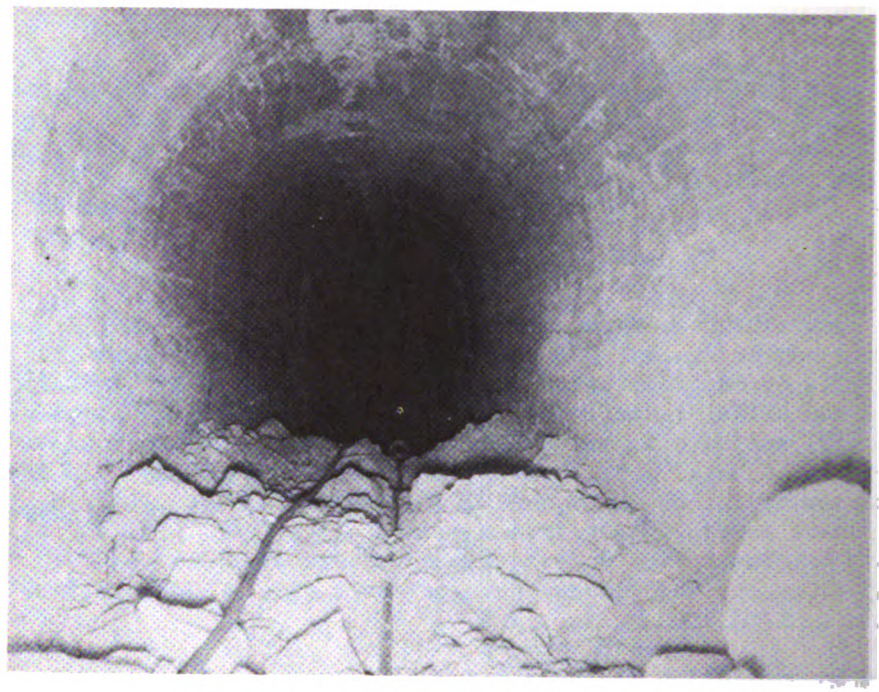

Figure 51. - Concreted slusher drift.

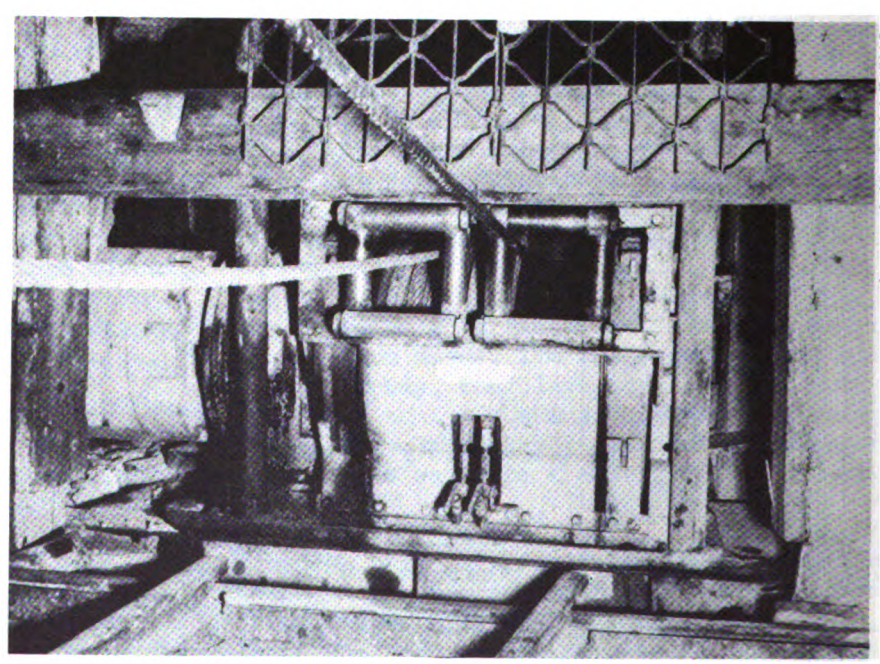

Figure 52. - Double-drum, 75-hp. electric slusher.

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Figure 53. - Sixty-inch folding scraper dumping ore into cars.

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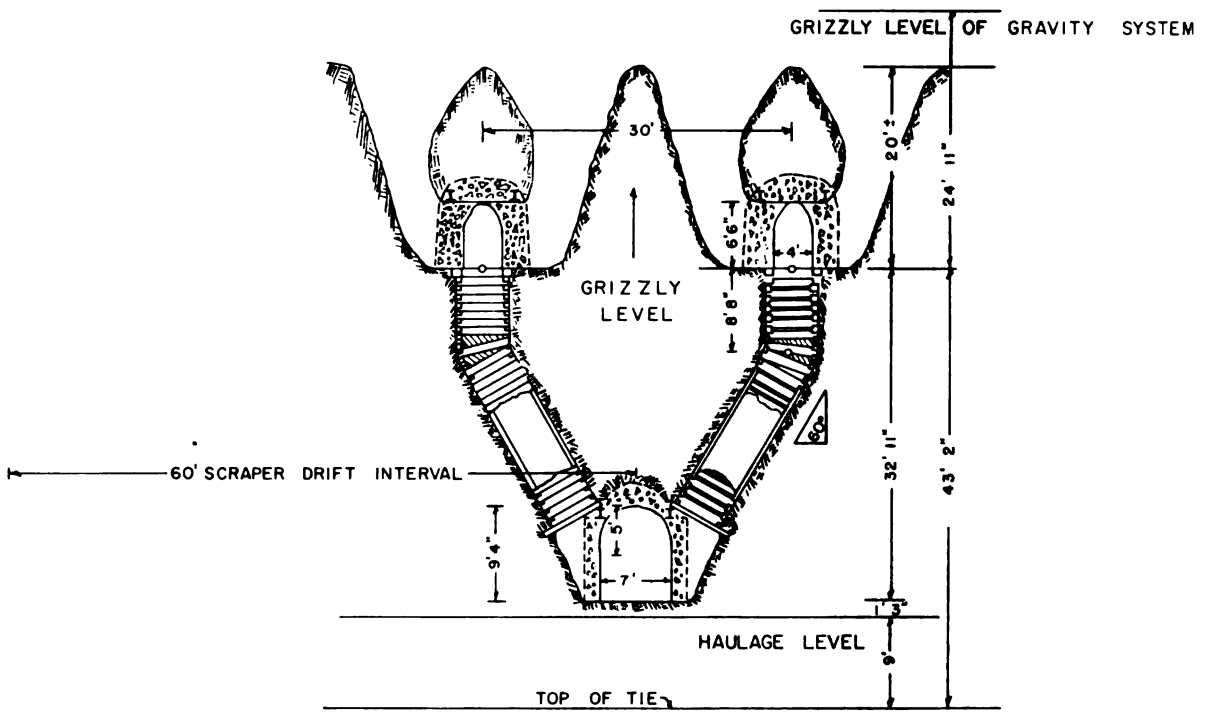
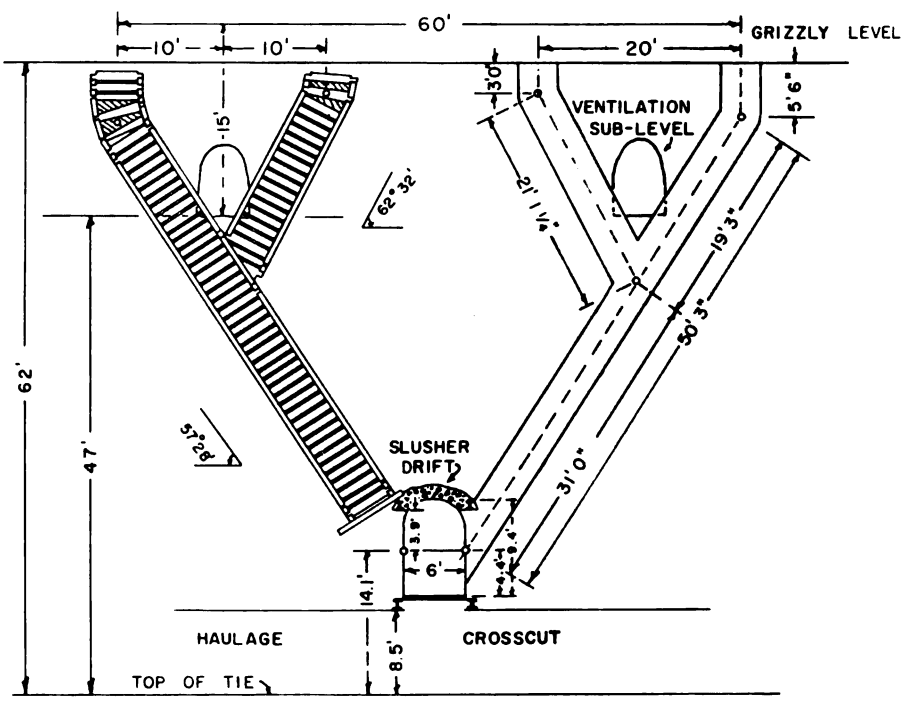


A. SLUSHER DRIFT PARALLEL TO GRIZZLY DRIFTS (MODIFIED RAISE DESIGN)



B. SLUSHER DRIFT TRANSVERSE TO GRIZZLY DRIFTS MODIFIED RAISE DESIGN WITH VENTILATION CROSSCUTS

Figure 54. - Block development, slusher-gravity system, sheet 2.

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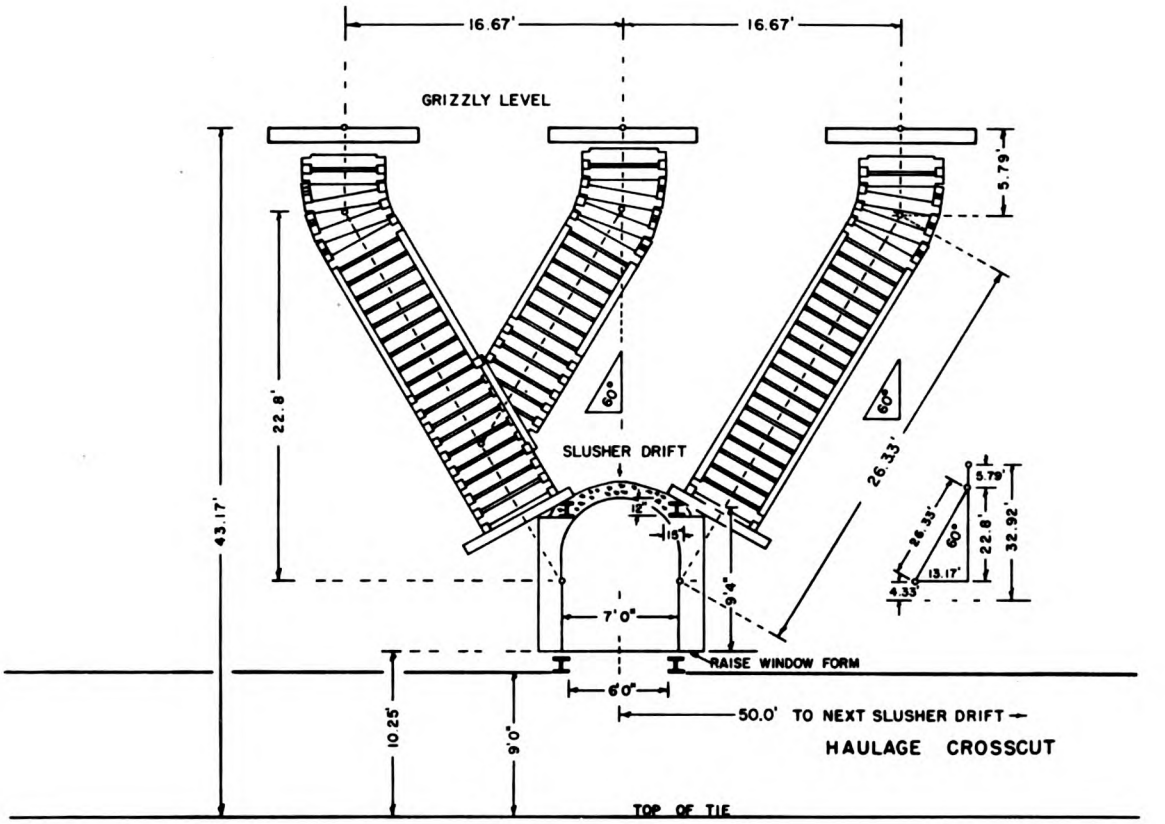
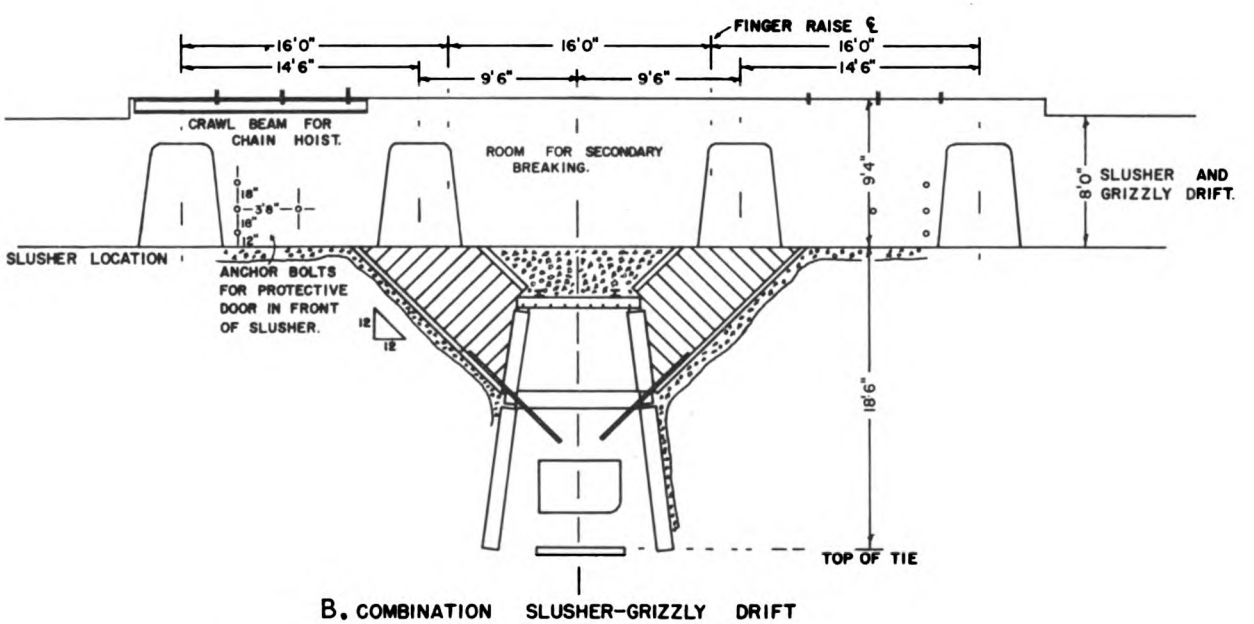


A. SLUSHER DRIFT, TRANSVERSE TO GRIZZLY DRIFTS
(THREE BRANCH RAISE DESIGN)



B. COMBINATION SLUSHER-GRIZZLY DRIFT

Figure 55. - Block development, slusher-gravity system, sheet 3.

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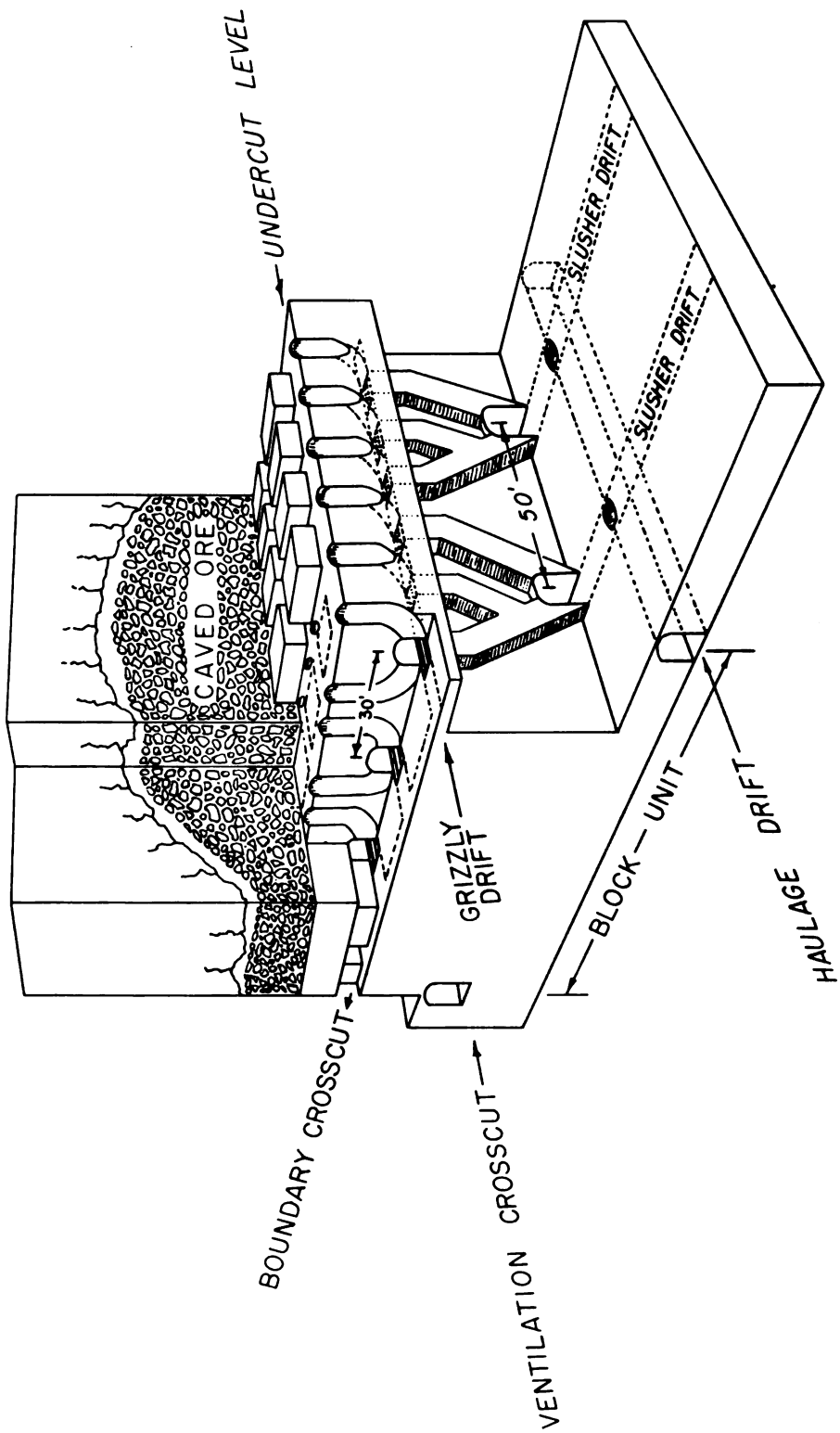

Figure 56. - Slusher-gravity block, three-branch raise design.

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In the final stage drawing is slowed considerably because of increased dilution from capping and adjoining exhausted blocks and the necessity of grade control by sampling. When the number of draw raises that satisfy grade requirements decreases, production slows.

At the Kelley mine the daily ore-drawing rate averages about 1 vertical foot per production day in terms of ore in place. In practice the rate varies from less than 1 to more than 2-1/2 ore-column-feet per day, equivalent to less than 20 to more than 50 tons of ore per finger per day. A faster rate than 2-1/2 ore-column-feet per day is being tried as long as uniform drawing makes it possible to maintain the original ore-waste contact angle in the block. Rapid drawing is practical chiefly because of semibroken character of ore bodies owing to previous mining, as evidenced by the ore broken to ore in place ratio discussed previously (p. 12 and table 3).

Production Planning and Control

Detailed plans for each zone, with sections for each panel showing the outline of mineralized rock by grade, are prepared by the geology department. Ore-reserve estimates, in terms of tons of ore and copper metal for each panel and block, are made by the mining-engineering and geological departments.

The required mine output is broken down by levels, zones, blocks, and grizzly lines and is scheduled on monthly, semimonthly and daily bases, depending on drawing conditions, transportation, and other operational factors.

Block development is scheduled at the same time, the main object being to prepare 1 ton of ore for each ton extracted; the required amount of undercut, in square feet, can be determined on the basis of production requirements and the tonnage available from panels and blocks. The progress of block development is kept by the engineering department on special charts showing the composite plan of grizzly and undercut levels and each block raise (figs. 57 and 58).

Careful daily records of tonnage drawn from each draw raise, grizzly panel, block, and zone are kept and compared with the estimates.

The daily stope draw reports filled in by shift bosses contain the number of cars drawn from each finger and from each grizzly drift (Appendix, form 1). The daily haulage report, filled in by the motorman, gives the number of cars drawn from each chute and raise for each trip (Appendix, form 2). The number of skips hoisted with ore is recorded automatically by a special attachment to the hoist. Knowing the weight factors for cars and skips, daily tonnage is readily computed and compared with tonnage recorded by belt weightometer.

The daily operating report for each zone is compiled by shift bosses. This report sums up the progress of block development in terms of contract labor, required materials, and the progress of ore drawing by listing day's pay labor for tapping and auxiliary work together with the number of cars produced (Appendix, form 3).

These basic production records, with assays of samples taken, furnish information for a series of tables (Appendix, forms 4, 5, and 6) and charts. The draw chart shows the daily production of individual draw raises and grizzly drifts compared with estimated tonnages (fig. 33). The records provide a graphic block history and, in a general way, indicate the relative position of ore and waste as drawing proceeds. They serve the management in planning ore drawing to secure uniformity for mill feed and other current production requirements.

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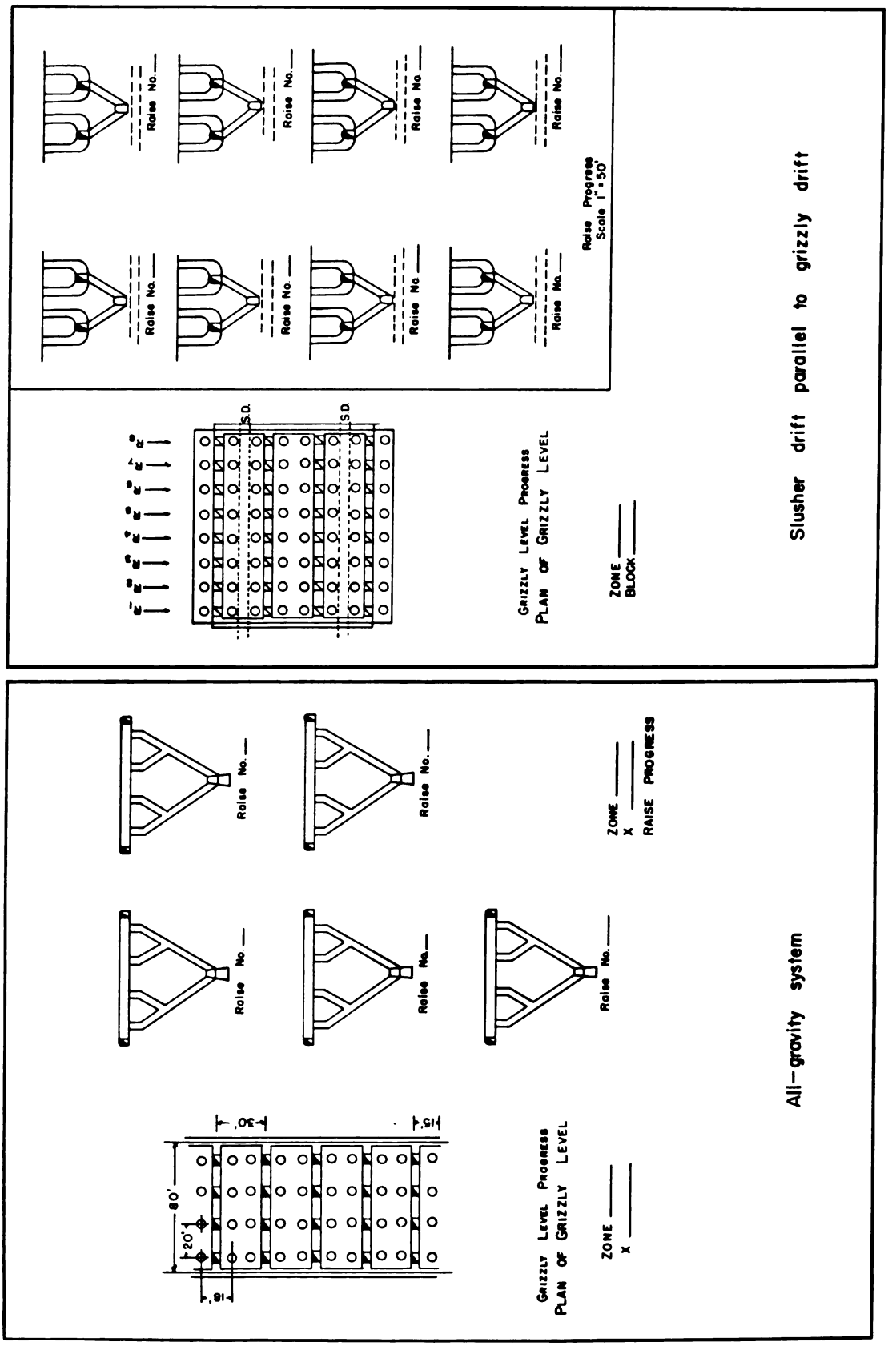

Figure 57. - Chart for following block development, list 1.

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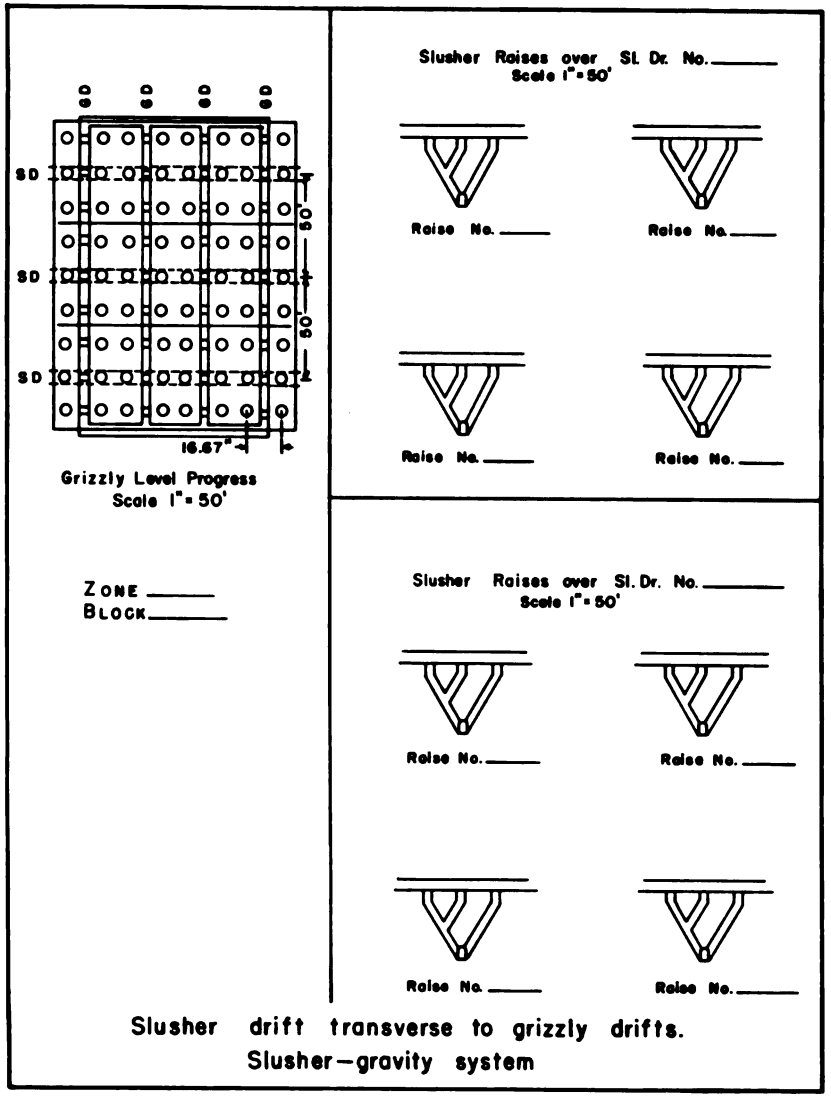
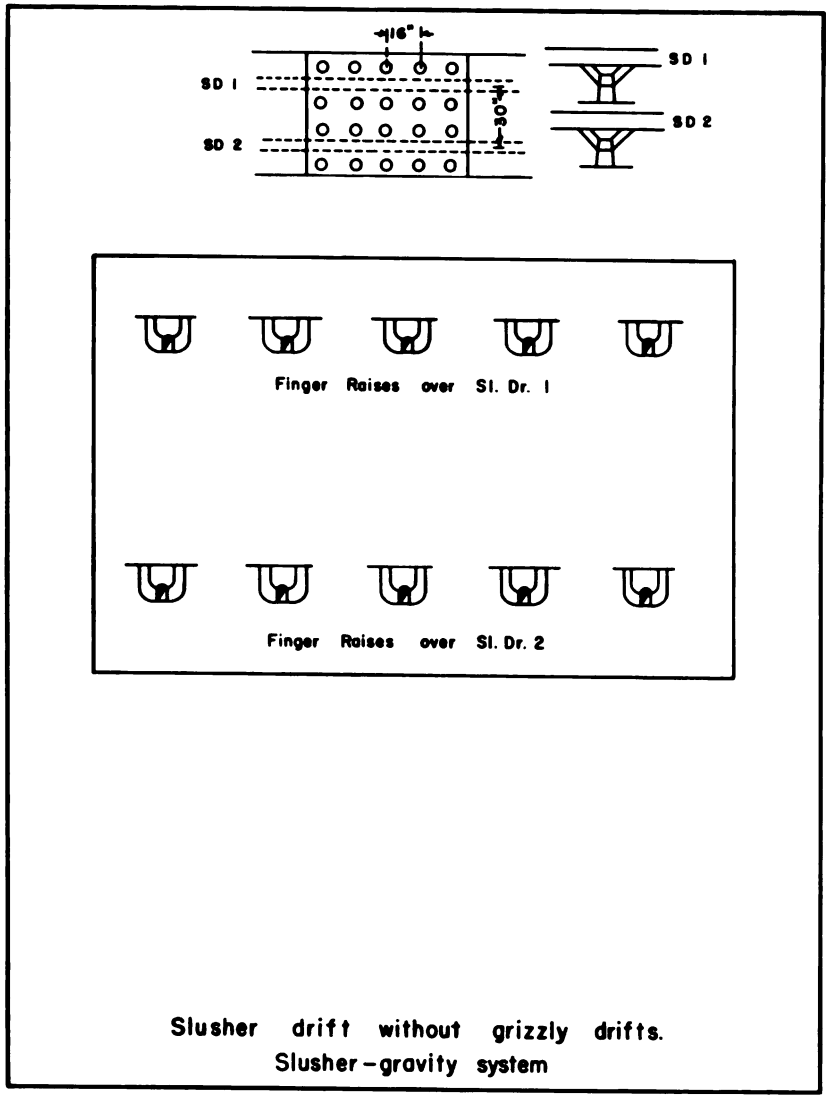


Slusher drift transverse to grizzly drifts.
Slusher–gravity system



Slusher drift without grizzly drifts.
Slusher–gravity system

Figure 58. - Chart for following block development, list 2.

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Individual block statistics kept on special forms show production tonnage from undercutting operations and ore drawn with grade determined by mine sampling and adjusted to smelter returns for any period of time (Appendix, form 7).

The progress report for the mine is made by mine management semimonthly (Appendix, form 8). It summarizes the status of development by zones and levels and block development by operations, stating the work completed in percentage of that scheduled, an estimate of reserves for individual blocks, and tonnage produced by sources and grade. The semimonthly report also shows the weighted average grade of production, determined on the basis of mine samples; this grade is compared with the composite mine assay of total and oxidized copper and with analyses of total and oxidized copper tonnage treated at Anaconda during the same period.

Quality Control

Grade is controlled by extensive and systematic ore sampling, namely, by interval channel sampling of all exploration and development workings, by grab sampling each car to make a composite sample at the dumping station for each trainload, and by sampling each finger during the critical period of drawing when dilution is suspected.

Channel and car grab samples are taken by the samplers under the supervision of the mine geologist. Finger samples are taken by the chute tappers under the supervision of draw bosses and occasionally for checking purposes by the sampler or geologist. All samples are crushed, mixed, and reduced to a size of several pounds underground (fig. 59).

The magnitude of the sampling is well-illustrated by the number of samples taken daily for the 12,500-ton output, namely, about 125 car grab samples and about 175 finger samples, a total of 300 samples, excluding channel samples.

Processing and quantitative analysis of samples are organized as a continuous stream, with a complete cycle of 24 to 36 hours. Rapid availability of the assays is an important prerequisite for scheduling and regulating the mine output and its average grade.

Measures Against Hung-Up Fingers

The most common way to free fingers choked by large fragments, timber, and scrap iron is by stick prodding, blowpiping, and blasting a packaged bomb detonated with a special cord attached to an electric cap-lamp battery. If these measures are unsuccessful and the raise is safe to enter, the finger may be stulled, drilled, and blasted. A push-feed drill may be set up in the grizzly drift to drill the packed material without entering the finger (fig. 48). Drill rods 16, 21, and 25 feet long are used. The holes are loaded with dynamite and blasted with an electric cap and Primacord-type fuse.

An 18-inch pneumatic chain saw is used by the crew to cut old timber into small blocks for easier handling. The timber may be reused, returned to the ore stream, or removed with scrap iron for later disposal by passing both down a raise to a supply car.

The ore-drawing crew, 4 to 7 men per shift for each block of grizzly level, consists of chute tappers and blasters. About 50 percent of a tapper's time is taken up in secondary breaking and 50 percent in freeing packs (figs. 60, 61, and 62).

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Figure 59. - Underground sampling layout on 600 level.

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Figure 60. - Chute tapper running rock from finger raise on grizzly level.

Figure 61. - Old timber coming through finger raise.

Figure 62. - Old 10-inch-column pipe coming through finger raise.

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Dilution

Due to Capping

The main source of dilution in caved ore is the waste capping above the ore column. Other sources are: Scouring, spalling, ripping off of the country rock, caving of the hanging wall, ore mixing with the waste at the boundaries of mined-out blocks, and ore mixing with filling.

In the initial phase of drawing caving action is restricted to the ore column, and there is virtually no dilution until sloughing and crushing reach the capping. Caving is possible only after some of the excess broken ore has been drawn. The ore tonnage available and the length of time required for this phase of drawing can be accurately estimated if the ratio of volumetric change of broken ore to ore in place is known. This ratio varies considerably for different ores. For Kelley-mine ore it is 1.25 and for porphyry-copper ore at the Inspiration mine about 1.60 (part 1, table 3). In other words, when 1 cubic foot of Kelley ore is broken its volume increases 25 percent compared with an increase of 60 percent for porphyry-copper ore. According to estimates, at least 20 percent of an ore column in place at Kelley, when broken, can be drawn without top dilution; this compares with 37.5 percent for porphyry-copper types of ore. Thus, at the same rate of drawing, caving will reach the top of an equal ore column in nearly half the time in the Kelley mine that it would be in a porphyry-copper deposit.

After disturbing the capping by removing 20-25 percent of the ore column, drawing the next ore with minimum dilution depends on the character of the cap rock and ore. Although even drawing to establish and maintain a broken ore-broken waste plane is the best practice, it does not eliminate the possibility of early dilution by channeling in a block disturbed by structures and by old workings. Ore may be diluted early also by loose, fine, iron-stained capping that works down through the broken ore and by channeling of waste owing to irregular drawing. In the latter case, pipes of faster moving ore are formed in broken ore first, then they work up and eventually reach the capping. Such channeling may be indicated by subsidence holes on the surface and by the color of ore drawn.

In general, after the capping is disturbed the dilution increases gradually with the quantity of ore drawn. In terms of the ore column this phase of drawing at the Kelley mine covers the range from 20 to 25 percent to 50 to 65 percent of the total and does not require systematic finger sampling.

In the final phase of drawing the ore becomes more and more mixed with the waste of the capping, as well as with waste from the country rock and from exhausted blocks. A consistent drop in ore grade is a characteristic feature, and to control drawing systematic finger sampling is required.

The rule adopted for sampling at the Kelley mine is to take a sample from each 50 tons of ore drawn from a finger. This is equivalent to about 2-1/2 feet of vertical column in an all-gravity system.

The minimum grade of ore, or rather the permissible degree of dilution, is determined separately for each case. The minimum permissible grade of ore depends on the grade originally estimated, on the cost of drawing, hauling, and hoisting, on the overall tonnage available, and on the grade of current mine production.

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Due to Country Rock

Dilution at the block boundaries may occur if the waste or below-grade country rock breaks and caves together with the ore. At the Kelley mine the boundaries of some ore zones are determined by sampling exploration workings and interpolating between levels. Mining of unprofitable ore is possible unless a uniform rate of drawing with an adequate system of sampling is followed.

Due to Filling Material

In most cases dilution due to ore mixing with rock, sand, and slime fill in the old workings can be estimated. If the volume of old workings is known from maps and sections, the influence of the fill on the grade of ore drawn can be evaluated with acceptable accuracy.

Dilution at the boundaries may be caused by the waste filling of exhausted blocks. The amount of such dilution depends on the degree of waste consolidation. Enough time is allowed for such consolidation to take place; drawing of bordering fingers is controlled by sampling.

Results of Extraction

More ore is drawn than was estimated. The results of drawing 34 blocks, a total of about 7,000,000 tons, show a tonnage extraction of 102.8 percent of that estimated and metal extraction of about 88 percent of that estimated. These compare favorably with block caving at other mines:

	Ore tonnage, percent of estimated	Copper-recovered tonnage, percent of estimated
Kelley mine, Butte, Mont.	102.8	88
Inspiration, Ariz.[1]	>110.0	>85
Ruth mine, Ely, Nev.[2]	104.0	87
Miami, Ariz.[2]	115.2	102.4
Miami, Ariz. (pillars enlarged)[3] ..	89.9	90.1

1/ Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.
2/ Peele, Robert, Mining Engineers' Handbook: 3d ed., vol. 1, sec. 10, p. 341.
3/ Still, J. W., Block Caving at Miami: Min. Cong. Jour., April 1955, pp. 89-92.

ROCK-BOLTING PRACTICE

Rock bolting has been used in Butte mines since 1939 and is now an accepted practice.[21 22] It is applied to keep ground self-supporting and to prevent sloughing and is replacing conventional timber with considerable success. Rock bolting has been standardized by the research department, and its instructions are carried out in detail by the miners.

In the Kelley mine rock bolting in haulageways has saved development time, decreased the cost of drifting and drift maintenance, and improved mining conditions and the efficiency of labor by providing better roof support and more work clearance

21/ Pollish, L., and Sandvig, R. L., Rock-Bolting Experience in the Butte Mines: Min. Cong. Jour., August 1952, pp. 50-52 and 123.
22/ O'Leary, V. D., Rock Bolt Developments in Butte Mines; Current Safety Topics in the Mining Industry: Trans. Nat. Safety Council, vol. 20, 1953, pp. 44-48.

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for miners and mechanical equipment by improving ventilation, reducing potential hazards, and eliminating fire hazards.

The resultant smaller cross section in haulageways has reduced the volume of excavation 22 to 27 percent and saved 25 percent of the time formerly required. Rock bolting was used successfully before the lining of the service adit leading from the mine yard to the Kelley shafts was concreted. It is employed before concreting underground stations and ore pockets and is used to support trolley line, pipelines, and ventilating tubing and to anchor sheave blocks.

The bolts are installed in a definite pattern determined by ground conditions. In 9- by 7-foot haulageways the bolts are set in rings spaced at 4- to 5-foot intervals; each ring consists of 5 bolts spaced 3 feet apart (fig. 27). Wooden headboards, steel plates, and mats are used to distribute the strength of the bolt. The bolts are 1-inch-diameter slot-and-wedge type and rated to withstand 18 tons of tension. Bolts are tightened with a 30-pound, air-driven impact wrench to a tension of about 10 tons; the wrench is capable of putting a strain of 13 to 16 tons on the bolts. Bolts are pull-tested with a hydraulic bolt tester to check for proper anchorage.

CONCRETING PRACTICE

Concreting is done by a special crew after excavations to specified size have been completed. Ground is supported by rock bolts, when possible, or by special steel sets. Prefabricated sectional wood forms are used for grizzly and slusher drifts (figs. 44 and 50). The length of drift section concreted at 1 pour is equal to raise spacing, which varies from 15 to 22.5 feet. The wood forms may be reused as many as 20 times.

Premixed concrete is delivered to the shaft or churn-drill hole collar, is transferred underground by pipelines, and is placed by a Pump-crete machine or pneumatic placer. An average pour, 22 to 30 cu. yd. in volume, requires about 4 hours.

Tests with prefabricated, sectionalized, collapsible, nontelescopic steel forms have been successful and are being used on an ever-increasing scale (fig. 22). The complete form unit consists of five 4-1/2-foot panels for concreting 22.5 feet of drift section in 1 pour. The central panel of the unit can be removed where a raise occurs and a lintel panel substituted. The complete unit is installed on a track-mounted car for easy transportation.

About 650 feet of grizzly drifts and 250 feet of slusher drifts are concreted monthly. This requires placing 1,500 cu. yd. of concrete. Cost of concreting is close to that of timbering.

VENTILATION AND AIR CONDITIONING

The problems of ventilation and the methods and equipment brought into use for their solution in Butte mines are discussed by Richardson in several publications.[23 24 25 26]

23/ Richardson, A. S., Ventilation of Butte Mines of Anaconda Copper Mining Co.: Trans. AIME, vol. 58, 1923, pp. 33-60.

24/ Richardson, A. S., Air Conditioning for the Ventilation of Butte Mines: Eng. and Min. Jour., vol. 139, No. 10, October 1938, pp. 29-34.

25/ Richardson, A. S., Metal-Mine Ventilation Notes: Min. Cong. Jour., July 1952, pp. 51-53.

26/ Richardson, A. S., Ventilation and Dust Prevention in the Butte Mines: Trans. AIME, vol. 141, 1940, pp. 268-275.

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At the Kelley mine good atmospheric environment is maintained in all workings by a well-planned, efficient, modern ventilation system. The magnitude of the daily mine production, the nature of the ore deposits, and the pecularities of the block-caving method forced simultaneous operations in multiple working areas and a large underground mine crew. These factors, and the need for special measures to control heat, owing to temperature rise with depth, humidity, air composition, smoke and dust, oxidation of ore and timber, and dormant fire and gas in old workings governed the design of the ventilation system.

Details of the Kelley-mine ventilation system are beyond the scope of this report. In general, it follows the modern practice of dividing the mine into independent ventilation districts, supplying large quantities of air, splitting the air current at each level and ore zone to bring the fresh air into each working place by the shortest possible route, and controlling dust at its source.

Air is supplied through the Kelley shafts with a system of exhuast fans installed at the ventilation shafts for each zone (fig. 63). Ventilation drifts are driven around each caving zone on the grizzly level and connected with exhaust fans and dust-collector units. These units installed to scrub the air and remove the dust are of two types: The electrostatic precipitator and the wet impact collector with water spray and glass baffle plates. Six collectors are installed underground, and at least six more are planned for future installation. Other modern equipment includes huge fans of as much as 300,000-cu. ft.-per-minute capacity and numerous auxiliary blowers operated with flexible tubing for ventilating long and dead-end development workings (fig. 64).

The system provides for sealing abandoned workings; installing special devices for controlling air current; proper design of airways, such as smooth lining; and building fireproof underground fan stations.

A new and effective design of ventilation tried for an all-gravity system of mining consists of specially driven ventilation drifts connecting the junction of the main raises below the grizzly level and joined to an air exhaust lateral or exhaust fan. The air current is downcast through the grizzly openings to the ventilation drifts. The system eliminates dust at the grizzly-level openings and provides fresh and cool air for chute tappers. It also provides better ventilation on haulage level by uprise air movement in the main raises.

The design and use of modern equipment permit the air flow to be controlled by mechanical and other means for individual working places. Constant supervision and inspection assure proper atmospheric conditions in all underground workings. On the whole, the system has increased the efficiency of workers considerably.

DUST PREVENTION AND CONTROL

Elimination of dust at the place of its origin is the goal of mine management. The accepted mining practice of wetting the working face, wetting broken ore after blasting, and wetting ore when drawing and loading is widely adopted. New antidust devices introduced by the company and used successfully include: Overhead ore-car sprinklers on the main haulageways operated by the train crew; tank trucks pulled by a motor for sprinkling haulageways; compressed-air and water-mist sprays; Ventube orifices producing a flat tabular ribbon of air of high velocity to insure positive displacement from the workings; and a track-cleaning machine for haulageways, laterals, and crosscuts.

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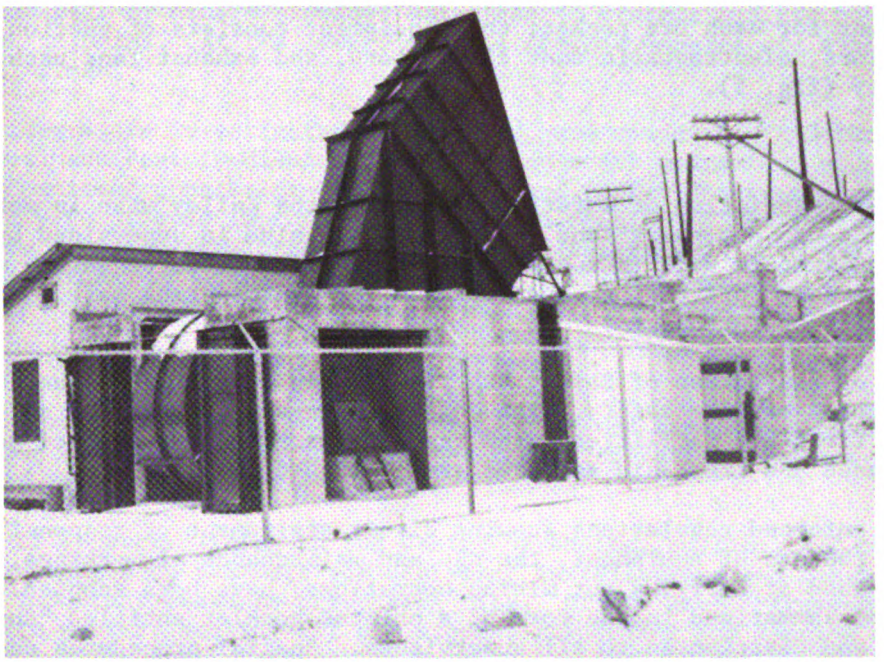

Figure 63. - Neversweat ventilation shaft—surface installation: 500-hp., 880-r.p.m., and 165,000-c.f.m.

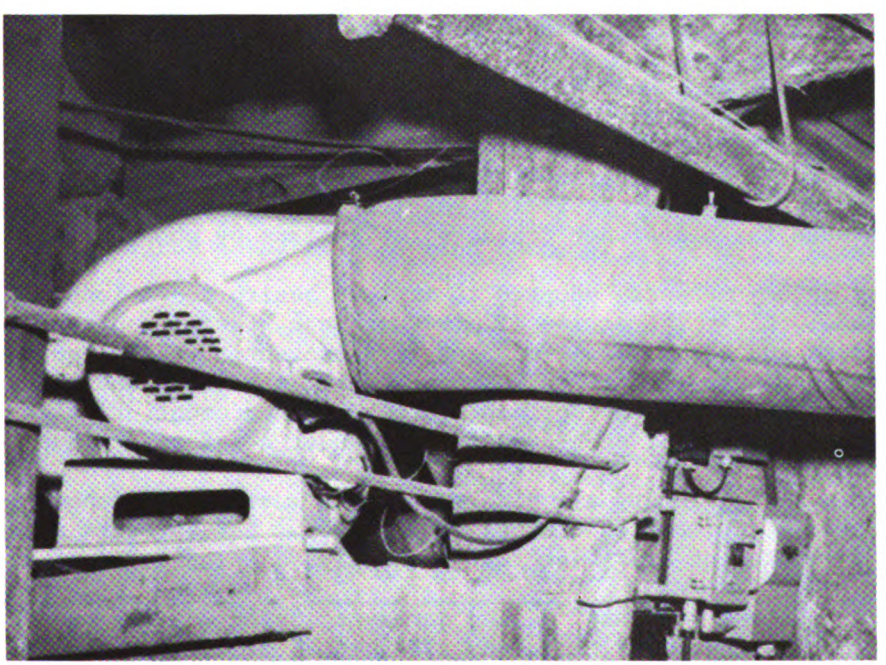

Figure 64. - Sixteen-inch, high-speed auxiliary blower with flexible tubing.

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Dust from dumping on the 600 and 1,300 levels is eliminated with specially designed equipment for each ore pocket; this equipment consists of suction hoods, wet impact collectors, electrostatic dust eliminators, and exhaust fans each 20,000-c.f.m. capacity (fig. 5).

SAFETY

A detailed description of safety practices at the Kelley mine is merited. As part of the Anaconda Co. operations, safety at this mine is supervised by the Bureau of Safety, organized in 1914 in recognition of the importance of safety measures, the value of accident prevention, and the need for coordinating safety efforts in mining and allied operations.[27/]

The work of the Bureau of Safety has resulted in much improved health conditions and better control of mine safety. The company maintains in Butte a safety station supplied with first-aid facilities, mine rescue apparatus, and trained personnel. Special ambulance and first-aid services are available at all times.

Strictly enforced regulations showing direct assignment of responsibilities and governing the use of equipment, the use and maintenance of workings, and the recommended method of carrying out mining operations, have been adopted. These regulations are summarized in the form of a printed booklet entitled "Supervisor's Safety Codes" and distributed to all supervisors. Special procedures dealing with fire control, fire fighting, and mine rescue operations have been worked out.

Training in safety practices is given to all new men employed.[28/] Each miner is furnished with an illustrated book of safety rules explaining the proper use of tools and recommended safety measures. First-aid training of miners is conducted regularly through a program recommended by the Federal Bureau of Mines. The Safety First campaign is carried out incessantly with safety bulletin-board posters, proper signs at working places, and individual and group discussions by the mine staff.

Constant supervision of all major operations by telephone, two-way radio, signal system, and in person helps promote adherence to safe practices. A daily inspection of workings is made by a specially assigned safety engineer. On the 600 level the dispatcher station is provided with a first-aid cabinet, stretcher, and four Pneolators for automatic artificial respiration. In the event a doctor or ambulance is needed the dispatcher contacts the surface by phone.

Compulsory safety measures adopted to improve health conditions and to prevent accidents include: Daily inspection of mine workings, checking of the temperature and humidity, dust and gas elimination, and fire prevention. All equipment is regularly inspected to assure that it is in good working condition and that it is operated in a safe manner only by authorized persons. Enforcement of safety rules is aided by reducing the size of each crew working under a safety-trained supervisor.

To prevent personal injury all underground men must provide themselves with hard hats, safety shoes, working clothes, and gloves. Safety belts and goggles provided by the management are inspected frequently. A plastic first-aid packet

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27/ Goodale, C. W., and Boardman, J. L., Bureau of Safety of Anaconda Copper Mining Co.: Trans. AIME, vol. 58, 1923, pp. 8-32.

28/ Bishop, L. F., Job Training at the Butte Mines of the Anaconda Copper Mining Co.: Trans. Current Safety Topics in the Mining Industry, Nat. Safety Council; Chicago, Ill., vol. 20, 1953, pp. 28-29.

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carried on the belt by miners provides materials to protect against possible infection of minor injuries. At the lamproom small ammonia ampules are available to miners at all times.

All accidents are investigated, classified, and recorded. A detailed report of each individual accident is discussed by the mine staff and safety engineers. A Mine Safety Committee consisting of management and representatives of the miners meets regularly to discuss improvements in health conditions and safety measures. In addition to accident rate records for the mine being used for comparison with those of other mines, an accident rate for each shift boss is kept in the office for each month and year.

FIRE PREVENTION

All fire-prevention measures are supervised by the fire superintendent of mines. In case of fire he also conducts the rescue operations and the fire fighting with a trained crew.

The use of concrete lining in the design of shafts, surface, and underground stations and underground chambers for equipment installations and other facilities are the most important measures preventing fire. Others are: Fireproofing all wood structures with gunite, sealing off old stopes, inspecting and keeping all electrical and mechanical equipment and cables in good working condition, strictly enforced regulations concerning blasting, forbidding smoking, proper storage of flammable supplies, and cleaning all workings of refuse.

Measures against spreading fire, gases, and smoke include fire doors installed in airtight concrete frames at all critical points, such as at the entrance of shaft stations and haulage intersections. In addition, air current may be controlled by reversible fans installed on the surface for ventilation.

To fight fires waterlines that extend to all working places for operating and drinking purposes are supplied at intervals with taps, nozzles, and hoses, airlines are convertible to water, and hand carbon dioxide extinguishers are placed at stations and near operating equipment.

Fire-prevention rules are enforced by mine management and the safety engineer. Fire watchmen are assigned to patrol the mine workings incessantly between shifts and on week ends.

SUBSIDENCE AND GROUND MOVEMENT

The north part of Butte adjoins the Kelley mine outside the zone of probable subsidence and ground movement. Within the zone of possible subsidence, shafts and buildings have been dismantled and their functions moved to other mining structures.

Subsidence is closely and continuously inspected. The surface is covered by a regular transit survey grid at 200-foot intervals showing accurate elevations for all coordinate intersections. All ground movements are recorded, and subsidence contour maps are drawn periodically.

Vein and fault structures are widespread and are expected to limit the effect of subsidence to the area closely surrounding the ground being mined. In any case, little damage to surface buildings is anticipated beyond the 45° line for the undercut level.

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ORGANIZATION

General Operating Staff

All company mining operations are directed by the general superintendent of mines with the assistance of the mining engineering, mining research, geology, and other departments.

The main duties of the general superintendent of mines are: (a) Organizing and directing the mine operating staff; (b) establishing and scheduling development, production quotas, and grade minimums for all mines; (c) providing mines with supplies and materials for uninterrupted operations; (d) coordinating the use of existing subsidiary facilities for servicing and maintenance (filling, transportation, ventilation, power, air, concrete); (e) designing mining installations; (f) controlling mine expenditures; and (g) enforcing operating instructions and safety measures.

The mining engineering department is responsible for: (a) Surveying - surface and underground; (b) planning, designing, and controlling mine development workings (level layout, haulage crosscuts, transfer raises) and block development (raises, grizzly, and undercut levels); (c) planning, and controlling construction of surface and underground facilities; and (d) measuring all contract operations for labor payment.

To increase the overall efficiency and to lower the cost of operations, the mining research department carries out systematic, continuous studies and appraisal of all phases of mining, engineering, and equipment performance. Projects for applied research are undertaken with the approval and cooperation of the mine management to assist with the development of new methods and practices or to meet changing mining conditions. The most important responsibilities of this department are: (a) Comparative efficiency studies of various mining operations regarding the use of equipment and materials; (b) development of new operating methods, simplification and improvement of accepted practices, working out new techniques and detailed instructions for the establishment of these practices, and demonstrating and training personnel in their use; (c) cost studies and analyses of previous operations and establishment of time and cost standards; (c) setting up a system of production and grade control; (f) training supervising engineering personnel; and (g) long-range planning.

Mine Management

The accompanying chart illustrates the organization of the Kelley mine (fig. 65). The management consists of an assistant general superintendent in direct charge of all mining operations and responsible to the general superintendent of mines; a mining engineer in charge of long-range planning, design, and construction specifications and engineering surveys who works in close contact with and under the supervision of the engineering department; and a geologist in charge of geologic documentation and sampling. All three have definite duties and responsibilities, and work as an operating team, drawing necessary services from all departments of the company. Ventilation, fire prevention, research, equipment, and safety measures are under constant inspection by specially assigned members of appropriate departments reporting their observations and recommendations daily, or at certain intervals, to the assistant general superintendent.

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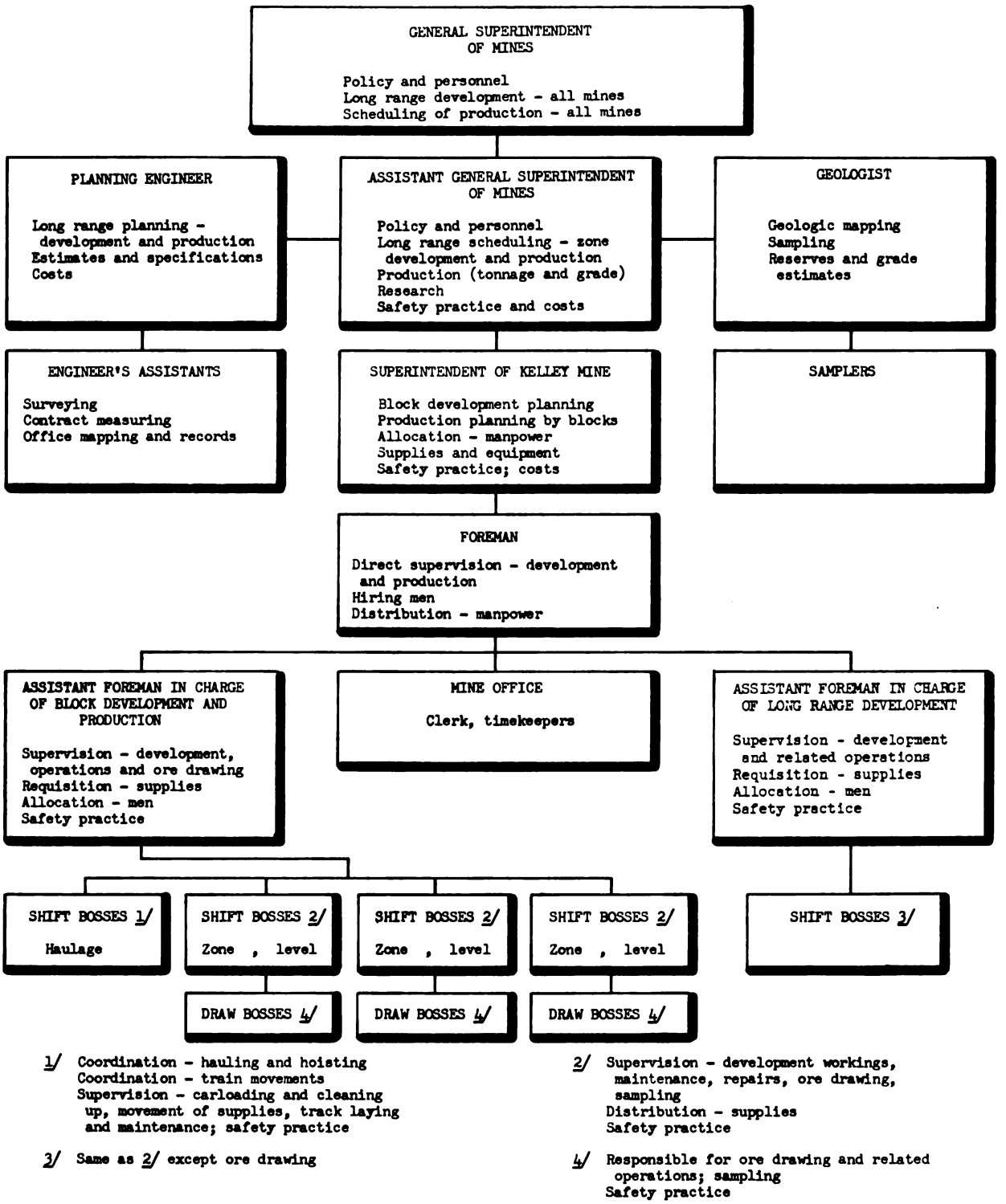

Figure 65. - Organization chart, Kelley mine.

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Mining operations are supervised by a mine superintendent assisted by a foreman with a staff of assistant foremen, shift bosses, draw bosses, and office personnel. Draw bosses on each shift are assigned to one or two blocks. These men assist the shift boss in ore drawing and maintenance work in their respective blocks.

Labor

The company used a contract system to increase the efficiency of labor. Most of the development work is being done under contract, with prices for each item based on units completed with the company guarantee of day's pay. Agreement with the union specifies additional pay for the afternoon shift and for the night shift.

The regular items paid by contract are: Breaking ground in shafts, drifts, crosscuts, raises and other headings - per linear foot or per cubic foot of volume; timbering - per cubic foot of volume or per set; rock bolting - per unit; and mucking and loading - per linear foot or per cubic foot of ground in place. Miscellaneous contract items are: Cribbing - per cubic foot; chute lining - per side; filling - per cubic foot; track installation - per double foot; installation of grizzlies - per unit; and concrete work - per unit of work.

Ore drawing, tramming, hoisting, and repairing are done on day's pay.

EFFICIENCY OF BLOCK-CAVING METHOD

Higher productivity of block caving as compared with other methods used for copper mining at Butte is well illustrated by ore output per manshift and labor requirements per ton in table 12.

The overall output per man-shift for the Kelley mine in 1954 reached nearly five times that of other mines at Butte. This was accomplished in spite of only 80-percent production of its projected capacity and continuing development in advance of extraction on a scale far exceeding normal requirements.

TABLE 12. - Comparison of mine efficiency by block caving with other mining methods in Butte district

	1952	1953	1954	1955 1/
Labor output, average:				
Tons of ore per man-shift:				
Block caving	9.37	15.20	15.65	16.24
Other methods	2.36	2.58	2.43	2.65
Labor requirement, average:				
Man-hours per ton of ore:				
Block caving	.86	.53	.51	.49
Other methods	3.39	3.10	3.29	3.02

1/ First 6 months.

MINING COSTS

Direct mining costs per ton of ore are not available for publication. A percentage breakdown of total operating expenses for the mine apportioned among the various cost items is given in table 13.

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This distribution illustrates the high cost of block development (36.2 percent) to mining proper (38.1 percent) and the place of labor cost in relation to supplies (69.1 percent and 30.9 percent). The table also emphasizes the cost importance of preparing slusher drifts (10.6 percent) and grizzly drifts (10.2 percent) in comparison with raises (2.4 percent) and undercut operations (2.1 percent) in block development and of tramming (20.9 percent), chute tapping (11.6 percent), hoisting (3.80 percent), ventilation (3.4 percent), and drainage (1.80 percent) in the overall cost of mine operations.

TABLE 13. - Cost distribution for mining operations, November 1955 (breakdown in percentage of total costs to each item of the mining operations, as provided by The Anaconda Co. in lieu of actual cost data)

	Labor	Supplies	Total
	Percent of total cost		
Block development:			
Level development	0.18	0.02	0.20
Zone haulage crosscuts	1.30	.60	1.90
Main raises	1.00	.60	1.60
Grizzly drifts	6.50	3.70	10.20
Finger raises	.60	.20	.80
Undercut level	.70	.40	1.10
Boundary weakening	.20	.10	.30
Slusher drifts	7.10	3.50	10.60
Repairs	2.00	.30	2.30
Undercut (pillar extraction)	.40	.60	1.00
Concreting	1.30	.60	1.90
Drilling	1.20	2.30	3.50
Tools	.40	.40	.80
Subtotal	22.88	13.32	36.20
Ore drawing, tramming, and hoisting:			
Chute tapping	10.40	1.20	11.60
Tramming	16.90	4.00	20.90
Hoisting	3.60	.20	3.80
Repairs	1.60	.20	1.80
Subtotal	32.50	5.60	38.10
General expenses:			
Supervision and office	7.00	0.10	7.10
Sundries and miscellaneous	2.70	5.70	8.40
Power electric	-	2.20	2.20
Power steam	-	.80	.80
Power air	-	.80	.80
Ventilation	2.60	.80	3.40
Drainage	.30	1.50	1.80
Fire prevention	.60	.10	.70
Primary crushing	.50	-	.50
Subtotal	13.70	12.00	25.70
Total	69.08	30.92	100.00

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APPENDIX

KELLEY MAIN SHAFT - PERTINENT DATA

Location: 1,500 feet northwest of Anaconda shaft;
1,000 feet northeast of the Central Butte mine machine and electric shop;
300 feet north of Butte Hill yard and Butte, Anaconda & Pacific Railroad track.

Elevation of collar: 5,970 feet above sea level.

Inclination: Vertical.

Depth: 2,731 feet.

Planned capacity: 15,000 tons per day.

Section: Rectangular.

Cross sectional dimensions: Outside - 38 feet by 9 feet; finished shaft - 35 feet 2 inches by 6 feet 9 inches.

Lining: Reinforced concrete (vertical rods on 12-inch centers, and horizontal rods on 6-inch centers).
Thickness, 16 inches (minimum).
Concrete per linear foot of shaft - 8 cu. yd.
Structural steel, per linear foot - 800 lb.

Number of compartments: 5

Dimensions of compartments:

Skip hoisting (4) - 6 feet 6 inches by 6 feet 9 inches.
Service: Manway, power cables, and pipes (1) - 6 feet 6 inches by 6 feet 9 inches.

Main working levels: 600 (elevation 5,258 feet), 1,300 (elevation, 4,462 feet)
2,000 (elevation 3,798 feet)

North Main Skip Hoist

System: Balanced skips.

Type: A first motion double-cylindrical drum with shaft directly connected to direct current rotors; each drum mounted loose on the shaft is driven by a radial, multiple tooth clutch; post brakes and clutches hydraulically operated; equipped with Ward Leonard magnetic electric hoist control and dynamic braking.

Motors: 3,000-hp. (rated), 600-v., 60-r.p.m., d.c.

Motor-generator: 3,500-hp., 2,400 v., 3-phase, 60-cycle synchronous motor, driving a 2,500-kw., 600-v., d.c., Ward Leonard generator.

Drum type: Grooved, 5,500 feet of cable capacity in three layers size: 12 feet diameter by 8 feet face.

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Cable size: 1-7/8 inches diameter.

Maximum hoisting speed: 2,250 feet per minute.

Maximum drum speed: 60 r.p.m.

Maximum rope pull: 63,000 lb.

Rate of acceleration: 2.5 feet per sec. per sec.

Time of acceleration: 15 sec.

Rate of deceleration: 3.15 feet per sec. per sec.

Time of retarding: 12 sec.

Calculated hoisting capacities:
600 level - 972 tons per hour
1,300 level - 694 tons per hour
1,600 level - 619 tons per hour
2,000 level - 559 tons per hour

Skips

Type: Automatic bottom-dump with rubber roller-bearing guide wheels instead of shoes.
Overall length - 24 feet 9 inches.
Weight, empty - 22,000 lb.
Volume - 252 cu. ft.
Weight of load - 14.5 tons.

South Main Skip Hoist:

Type: A first motion, double-cylindrical drum with shaft directly connected to the motor rotor; the drums, mounted loose on the shaft, are driven through friction-type clutches; clutches and post brakes are hydraulically operated.

Motor: 2,250-hp. (rated), 600-v., 60-r.p.m., d.c.

Motor-generator: 2,500-hp., 2,200-v., 3-phase, 60-cycle, 514-r.p.m. synchronous motor, driving a 1,700-kw., 600-v., d.c. Ward Leonard generator that supplies current to the hoist motor.

Calculated hoisting capacities:
600 level - 680 tons per hour
1,300 level - 550 tons per hour
1,600 level - 491 tons per hour
2,000 level - 444 tons per hour

Skips

Type: Similar to those operated by north main hoist.

Overall length: 22 feet, 9 inches.
Weight, empty: 19,000 pounds.
Volume: 200 cu. ft.
Weight of load: 11.5 tons.

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KELLEY AUXILIARY SHAFT - PERTINENT DATA

Location: 142 feet south of main shaft.

Elevation of collar: 5,960 feet above sea level.

Inclination: Vertical.

Depth: 2,162 feet.

Section: Rectangular.

Cross-sectional dimensions: Outside minimum, 20 feet 4 inches by 9 feet 8 inches.
Finished shaft, 17 feet 8 inches by 7 feet 0 inches.

Lining: Concrete.
Thickness, 16 inches (minimum).
Concrete divider between compartments: 36 by 8 inches.

Number of compartments: 2.

Dimensions of compartments: Cage compartment (1)-14 by 7 feet.
Pipe, manway, and counterweight compartment (1)-7 by 3 feet.

Service cage: Double-deck capable of accommodating 90 men and rolling stock 12 feet 6 inches by 5 feet.

Dimensions: 5 feet 3 inches by 13 feet 0 inch (inside).
Height: 23 feet (from draw bar to bottom).
Weight: 19,000 lb.
Capacity: 90 men at a time.

Main levels: 600, 1,300 and 2,000.

Order of development: By raising from 600, 800, 1,300 and 2,000 levels.[29/]

Shaft timber: 10 by 10 inches, finished inside and outside; placed to become forms for the pouring of the monolithic concrete.

SURFACE INSTALLATIONS

Skip ore bins: Capacity - 1,400 tons.
Diameter - 30 feet.
Construction: Riveted steel with conical bottom.

Primary crushing plant: Apron feeder: Width - 4 feet.
Length - 15 feet.
Motor drive - variable speed.

Screen: Type - vibratory.
Size - 5 by 12 feet.
Holes - 6 inches.

29/ Rood, A. D., Why Anaconda "Raised" the Kelley Supply Shaft: Eng. and Min. Jour., May 1955, pp. 85-89.

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Primary crushing plant (Con.): Jaw crusher: Size - 36 by 25 inches. Setting - 6 inches.

Conveyor belt: Width - 42 inches. Length - 320 feet. Speed - 434 ft./min. Inclination - 16°.

Weighing meter: Merrick weightometer.

Ore bin: Capacity - 7,500 tons.
Length - 208 feet.
Width - 20 feet.
Height (above loading tracks) 54 feet.
Construction: Welded steel construction with an "A"-shaped bottom.

Cross-belt conveyor: Length - 250 feet.
Width - 36 inches.
Speed - 465 ft./min.
Type: Automatic, self-propelled, self-reversing belt tripper.

Loading facilities: Air-operated guillotine-type gates; loading - on each side of the bin by gravity.

Railroad facilities: Car capacity - 55 tons.
Train length - 42 cars.
Train capacity - 2,300 tons.

HEADFRAME AND HOISTHOUSE

Permanent Headframe: Base width - 82 feet.
Base length - 100 feet.
Height - 177-1/2 feet.

Construction: Steel.

Sheave wheels: 4.
Sheave height above collar - 160 feet; diameter - 12 feet.

Hoist House: Location: 300 feet east of shaft.

Dimensions: Length - 148 feet.
Width - 58 feet.
Height - 30 feet.

Construction: Steel frame building with corrugated-steel-sheet walls, rested on 24 inches of reinforced concrete mat.

Idler towers: Sheaves, diameter - 36 inches.

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UNDERGROUND FACILITIES

Ore pockets	600-level
Bins, number	2
Capacity, each	500 tons.
Form	Circular.
Diameter	14 feet.
Lining	60-pound rails concreted in place vertically.
Grizzly: Material	90-pound rails.
Bar spacing	24 inches.

Locomotives	600-foot level	Intermediate levels
Type	10-ton trolley units equipped with air brakes and sanders.	6-ton battery units.
Hp. of motors	120 on 2 axles.	
Weight	12 tons.	
Wheel base	5 feet 4 inches.	
Speed (full load)	10.7 miles per hour.	

Ore cars		
Type	Granby side-dump, with automatic couplings.	Granby side-dump.
Capacity	115 cu. ft.	57
Capacity ore (approx.)	5 tons.	2-1/2
Dimensions: Width (outside)	5 feet.	3 feet 5-1/2 inches.
Height (above top of rail)	5 feet 3 inches.	4 feet 6-1/2 inches.
Overall length (center to center, coupled)	10 feet.	9 feet 6 inches.
Inside length	7 feet 6 inches.	7 feet.
Material	Steel.	Steel.
Wheel base	3 feet 6 inches.	3 feet.
Wheel diameter	16 inches.	14 inches.
Type of bearings	Tapered roller.	Tapered roller.

Other cars:

Bench car closed, seating 12 men.
Wood-lined cars for explosives.
Flat cars for timber.
Toilet cars.

Mine track:		
Gage	36 inches.	18 inches.
Weight of rails	60 lb.	40 lb.
Tie dimensions	6 by 8 by 64 inches.	6 by 8 by 42 inches.
Tie spacing	18 inches.	36 inches.
Radius of minimum curve	75 feet.	40 feet.
Grade	0.3 percent.	0.3 percent.
Switches	Point or split. (frogs are interchangeable)	Point or split.

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Size (57 or 115)

STOPE DRAW REPORT – CARS 115

DATE _____ ZONE 9-600

SHIFT 8:00 A.M. LEVEL 600 X- 7

SIGNED John Doe

		1	2	3	4	5	
	N	10	12	8	15		
GD	1						84
	S	8	10	12	9		
	N	8	12	10	8		
GD	2						70
	S	10	8	6	8		
	N	10	10	12	8		
GD	3						80
	S	12	8	8	12		
	N	10	8	10	8		
GD	4						76
	S	12	10	8	10		
	N						
GD	____						
	S						
	N						
GD	____						
	S						
TOTAL		80	78	74	78	____	310

Form 1. - Stope draw report.

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10M-6-55-STANDARD

HAULAGE REPORT – CARS

Zone 9 Level 400 X- 9 Date Shift 8:00 A.M.

Motorman Henry Smith Size of Car (57 or 115) 57

Trip Raise

	1		2		3		4		5		6		7		8		9		10		11		Total
	E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W	
1						6	4																10
2								6				4											10
3							2	4			4												10
4						6	4																10
5						6	4																10
6						6			4														10
7								6			2	2											10
8								10															10
9						6	4																10
								4															10
18						6						4											10
19						6						4											10
20						6					4												10
21					5			5															10
22							6				4												10
23								10															10
24								10															10
Tot.					45	48	26	77	4		16	24											240

Form 2 Haulage report

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Left half

OPERATING Date October 4, 1955 — Day Shift: Smith — Night Shift: Green — Asst Foreman: White — Boss' Beat: Zone 1-1300 — Checked: — Form 3-4-54-5M.

Contract No. and Place	Shift	Account	Labor D.L.	Labor D.F.	Cars Ore Reg.	Cars Waste H	Cars Waste F	Rd Sq Rod Posts	Pound Square Segment Caps	Pound Square Segment Girts	Circle "O" All Used Material Lagging St.	Lagging 5'4" 2"	Lagging 5'4" 3"	Lagging 7'2" 2"	Lagging 7'2" 3"	Braces	Cribs	Explosives Used	Explosives Rec'd From Prev.	Explosives Rec'd From Stor.	Drill Type	Drill No.
606	D	01-03	2								2	1			4							
1-13-21CWGD2	N	13-10C	2		5							3				1						
607	D	01-05	1		15													135	12		JL	3
1-13-21CWUCDR	N	13-01	2		12						4			7								
607	D	01-04	1														8	62	9		P	
1-13-21 CW FR	N	13-01																				
616	D	01-01	2		2			2	1	2		6	5					160	17		JT	3
1-13-X 18	N	13-01	2		7	3-¾"x6' Rock Bolts												127	13		JT	3
622	D	01-14	2			6-¾"x6' Rock Bolts 4												35	6		JL	2
A1345XSW	N	13-01																				
636	D	01-03	2		4	4-1"x6' Rock Bolts					11							47	5		JL	2
1-13-24CWGD4	N	13-01																				
640	D	01-07	2		2													117	14		JL	3
1-13-24CWSLDr	N	13-01	2		9	7-¾"x6' Rock Bolts															JL	3
812	D	01-14	2															210	23		P	6
1-13-21AW VR	N	13-01	2		11			6	3	4		7	13									
X-12	D				246																	
Cove Rock	N				258																	

Right half

DAYS PAY LABOR

REMARKS	Place	01-01 Haulage & Cars	01-03 Grizzly Raises	01-04 Grizzly Levels	01-05 Finger Raises	01-06 Undercut	01-07 Slushers	01-08 Bonnet Work	01-09 Chutes	01-10 Chute Equip.	01-11 Vent.	01-12 Timber	01-13 Drill Maint.	01-14 Tram & Dump	01-15 Hoist	01-16 S.S.P.L.	01-17 Drills	01-20 Tools	01-28 Misc.	01-30 Fire	Misc. Explos.	Sec. 2 Timber
	1300								2	6	1			3	1		1	1			1	
	Level								2	6				3	1					1	1	

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57 Cu. Ft. Cars Hauled and Dumped into Transfers
Grab Sample from Each Car at Dump
Kelley Mine
Month of OCTOBER 1955

	Zone 9 to Transfer Raise D-689 (Hauled on 400 Level)									Zone 9 to St. Law Transfer Raise (Hauled on 400 Level)									Zone 11 to Transfer Raise E-603 (Hauled on 500 Level)								
	X-9	X-11	X-13	X-14	Dev.	Daily Total	Accumulative			X-13	X-14	X-	X-	Dev	Daily Total	Accumulative			X-2	X-8	X-	X-	Dev	Daily Total	Accumulative		
	Cars	Cars	Cars	Cars	Cars	Cars	Cars	Cars %	% Cu	Cars	Cars	Cars	Cars	Cars	Cars	Cars	Cars %	% Cu	Cars	Cars	Cars	Cars	Cars	Cars	Cars	Cars %	% Cu
1	430	408	60	135	37	1614				420	124			—	544	544			400	224			40	664	664		
2																											
3	402	302	—	122	20	1459	3073			367	246			—	613	1157			429	270			8	707	1371		
4	370	352	140	83	14	1693	4766			574	160			—	734	1891			396	192			16	604	1975		
5	410	418	118	75	3	1700	6466			490	153			33	676	2567			407	212			8	627	2602		
6	450	390	72	99	6	1515	7981			316	182			—	498	3065			429	125			53	607	3209		
7	470	390	112	104	32	1765	9746			460	183			14	657	3722			440	194			17	651	3860		
8	370	372	96	100	22	1572	11318			350	234			28	612	4334			422	120			18	560	4420		
9																											
10																											
11																											
12																											
13																											
14																											
15																											
1-15 Inc.																											
16																											
17																											
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22																											
23																											
24																											
16-24 Inc.																											
25																											
26																											
27																											
28																											
29																											
30																											
31																											
25-31 Inc.																											
Total																											

No 372 - File 184

115 Cu. Ft. Cars Hauled on 600 Level to Kelley Shaft
Grab Sample from Each Car at Dump
Kelley Mine
Month of OCTOBER 1955

	Zone G										Zone 8										Zone 9 - 600 Level										Zone 9 400 Level Transfer		Zone 9 600 Level Transfer		Zone 11 200 Lot Transfers		All 600 Development		Total				
	X-		X- 5		X-		X- 13		X-		X-		X- 18N		X- 18S		X-		X-		X-		X- 7B		X-		X-		X-								Daily		Accumulative			
Date	Cars	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Total	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Total	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Total	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Cars	% Cu	Cars	Cars %	% Cu
1			306				94				400		119		169						288		230								230		439		315		308		52		2032				2032		
2																																												
3			523				106				629		186		156						342		346								346		399		328		374		74		2492		4524		
4			494				154				648		182		163						345		224								224		423		378		337		87		2442		6966		
5			406				85				491		117		86						203		296								296		484		363		352		115		2304		9270		
6			321				79				400		137		50						187		274								274		396		282		312		88		1939		11209		
7			424				96				520		160		122						282		299								299		452		264		336		86		2239		13448		
8			391				100				491				42						42		361								361		439		345		336		92		2106		15554		
9																																											
10																																											
11																																											
12																																											
13																																											
14																																											
15																																											
1-15 Inc.																																											
16																																											
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24																																											
16-24 Inc.																																											
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26																																											
27																																											
28																																											
29																																											
30																																											
31																																											
25-31 Inc.																																											
Total																																											

No 342 File 180

Forms 4, 5. - Zone haulage reports for 57-cu. ft. cars and 115-cu. ft. cars, respectively.

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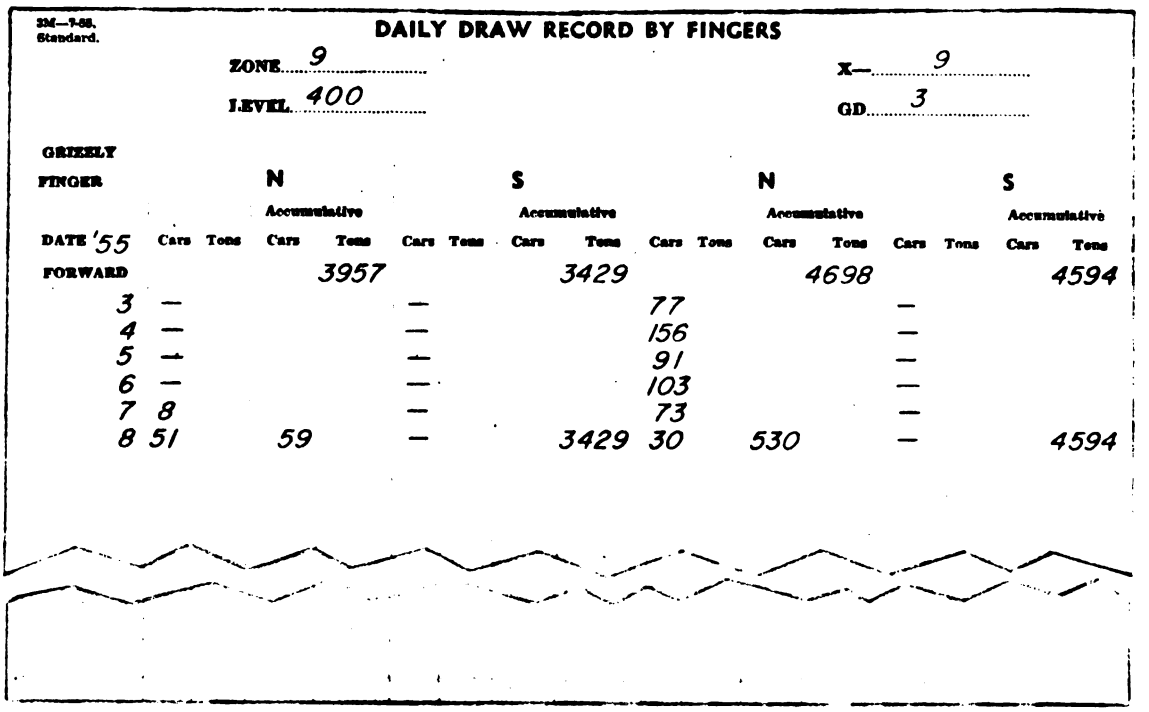


2M—7-55. Standard.

DAILY DRAW RECORD BY FINGERS

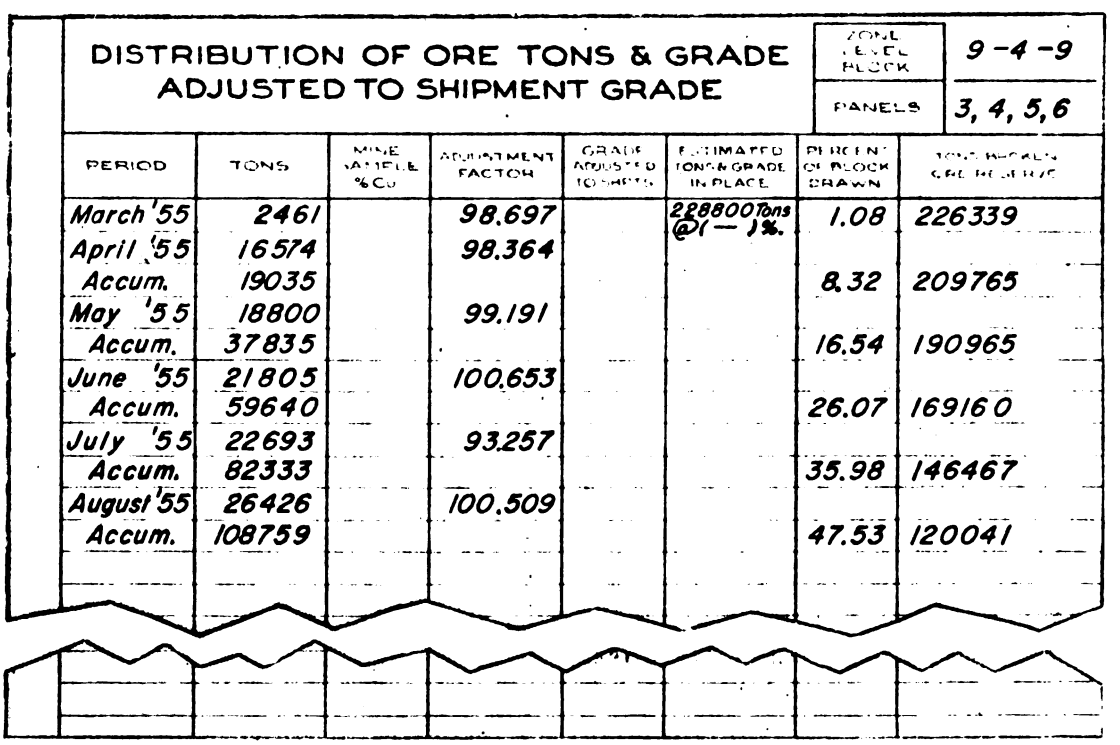
ZONE 9 X— 9

LEVEL 400 GD 3

GRIZZLY

FINGER	N				S				N				S			
			Accumulative				Accumulative				Accumulative				Accumulative	
DATE '55	Cars	Tons	Cars	Tons	Cars	Tons	Cars	Tons	Cars	Tons	Cars	Tons	Cars	Tons	Cars	Tons
FORWARD				3957				3429				4698				4594
3	—				—				77				—			
4	—				—				156				—			
5	—				—				91				—			
6	—				—				103				—			
7	8				—				73				—			
8	51		59		—		3429		30		530		—			4594

Form 6. - Daily draw report by fingers.



DISTRIBUTION OF ORE TONS & GRADE ADJUSTED TO SHIPMENT GRADE

ZONE LEVEL BLOCK 9 -4 -9

PANELS 3, 4, 5, 6

PERIOD	TONS	MINE SAMPLE %Cu	ADJUSTMENT FACTOR	GRADE ADJUSTED TO SHIPMENT	ESTIMATED TONS & GRADE IN PLACE	PERCENT OF BLOCK DRAWN	TONS BROKEN ORE REMAINS
March '55	2461		98.697		228800 Tons @1 — 1%	1.08	226339
April '55	16574		98.364				
Accum.	19035					8.32	209765
May '55	18800		99.191				
Accum.	37835					16.54	190965
June '55	21805		100.653				
Accum.	59640					26.07	169160
July '55	22693		93.257				
Accum.	82333					35.98	146467
August '55	26426		100.509				
Accum.	108759					47.53	120041

Form 7. - Block draw record.

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KELLEY MINE

I STATUS OF DEVELOPMENT — PROGRESS CHART

ZONE – LEVEL																	
HAULAGE CROSSCUT																	
RAISE PANEL NUMBERS IN BLOCK																	
ESTIMATED TONS IN BLOCK																	
ESTIMATED GRADE IN PLACE (% Cu)																	

PERCENT OF BLOCK DEVELOPMENT WORK COMPLETED

HAULAGE CROSSCUT																	
PONY SETS																	
MAIN RAISES																	
GRIZZLY LEVEL																	
FINGER RAISES																	
UNDERCUT LEVEL																	
BOUNDARY WEAKENING																	
UNDERCUTTING																	
UNDERCUTTING SCHEDULED TO REPLACE BLOCK																	

II STATUS OF PRODUCTION, PERIOD TO INCLUSIVE, 195 (WORKING DAYS)

SOURCE	ZONE LEVEL	HAULAGE CROSSCUT	RAISE PANEL NUMBERS	TONS PRODUCED	AVERAGE TONS PER DAY	AVERAGE GRADES % TOT. Cu.	% OX. Cu.	OZ. Ag.	% DRAWN OF ESTIMATED BLOCK TONNAGE	REMARKS
CAVING						a				
						a				
						a				
						a				
						a				
						a				
						a				
						a				
						a				
						a				
CAVED ORE TOTAL						b				
DEVELOPMENT ORE, ALL ZONES						b				
TOTAL MINE PRODUCTION						b				
TOTAL ORE TREATED TO INCL. 195						c	c	c		

a MINE CAR GRAB SAMPLE
b COMPOSITE MINE SAMPLE
c ANACONDA REDUCTION SAMPLE

INT.-BU. OF MINES, PGH., PA. 7676

Form 8. - Monthly progress report.

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